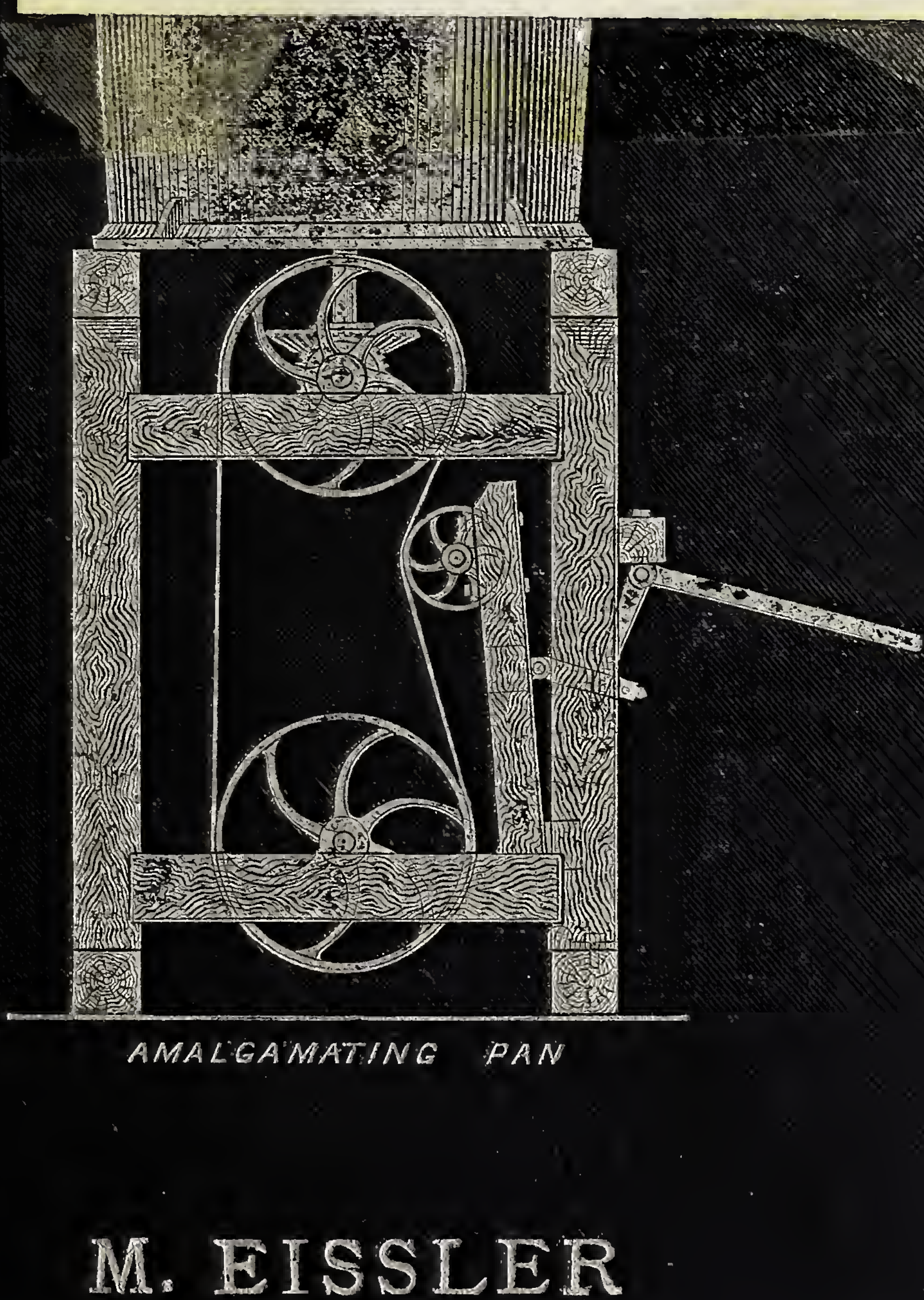
THE METALLURGY OF SILVER



AMALGAMATING PAN

M. EISSLER

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METALLURGY OF SILVER

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A PRACTICAL TREATISE

On the Amalgamation, Roasting, and Lixiviation of Silver Ores

INCLUDING THE ASSAYING, MELTING, AND REFINING OF SILVER BULLION

BY

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THIRD EDITION

With One Hundred and Fifty Illustrations.

Capio Lumen

LONDON
CROSBY LOCKWOOD AND SON
7, STATIONERS' HALL COURT, LUDGATE HILL
1896

PREFACE.

The object of this volume is to give a practical outline of the Metallurgical Treatment of such Silver Ores as are adapted to Amalgamation—that is, of ores whose character allows them to undergo treatment with quicksilver, with which metal silver forms an amalgam.

The processes of amalgamation may be classified as the Wet Process and the Dry Process. Ores which require no preliminary chemical preparation are adapted for direct amalgamation by the wet process. Ores which have to be treated by the dry process have to be crushed dry, and are roasted before being submitted to amalgamation—treatment which involves some very delicate manipulation and skill.

Professor Clarence King (in one of his Reports to the United States Government, to which, as will be mentioned presently, I have to acknowledge my indebtedness) has given an excellent account of amalgamation by the wet process; and from this report I extract a description of the Washoe process, as practised on the Comstock lode.

I then follow with a description of the application of the Washoe process in working ores from the Golden Chariot Mine in Idaho, as practised by me when in charge of the works.

In the description of the dry process I give my experience, at various mines, when dealing with such ores as require roasting, and I describe in detail the metallurgical operations at the works of the Mineral Hill Silver Mines Company, which were in my charge from 1870 to 1873; and although for several months I followed there the system adopted by my predecessor, who roasted the ore in a Stetefeldt furnace before amalgamation, I subsequently abandoned the dry process, as I found that the ores could be more profitably worked by the Washoe process, and I was thus able to effect a saving to the company of many thousands of pounds sterling every year.

For the roasting of ores many new furnaces have from time to time been introduced, which have greatly contributed to the development of the silver-mining industry by cheapening the cost of treatment, and thereby rendering productive mines which otherwise would have lain dormant.

Of late years a new science has been developed in the treatment of silver ores which cannot be worked profitably by amalgamation, and are too poor in lead, or of such character as not to smelt. I refer to the Lixiviation Processes, which are being successfully carried out in many localities. I have gathered in this volume the latest information

obtainable on this very important branch of Silver Metallurgy, as no doubt in the near future the lixiviation of ores will be more generally practised. Indeed, there is open here for inventors and investigators a very wide field, while success will mean large returns in money.

In describing in the following pages the several methods of treatment, full particulars, with illustrations, of the machinery employed for crushing, grinding and amalgamating are given.

The various processes included in Concentration of argentiferous minerals, by sizing, jigging, buddling, and vanning, will also be found fully explained and illustrated. Although concentration has not as yet been extensively introduced in the metallurgy of silver, I have devoted a portion of this volume to the subject, as I am convinced that in the near future large bodies of low-grade refractory silver ores will be worked by this system, recent experiments at mines having demonstrated that it pays better to concentrate than to submit a large quantity of unreduced ore to roasting, amalgamation, or lixiviation direct.

In this matter I have freely drawn on the experience and methods of English and German metallurgists, who have so largely contributed to bring concentration to its present state of perfection in the tin, lead, and copper districts; and I am more especially indebted to the interesting and detailed descriptions of some of the machinery employed which are given in the standard work of the late Mr. Robert Hunt, F.R.S., on "British Mining;" to

Messrs. Rittinger & Goetschman's "Aufbereitungskunde;" and to the Reports to the United States Government on the Mining Industry of the Pacific Coast, which were prepared several years ago by Professor R. W. Raymond and the eminent American geologist, Mr. Clarence King. I am also indebted to the kindness of Messrs. Fraser & Chalmers, of Chicago, of Mr. Krom, of New York, and of Messrs. Commans & Sons, of London, for specifications and drawings of improved modern machinery. Information as to the Russell process and the working of ores at Tombstone, in Arizona, and at the Ontario Mine, in Utah, has been gleaned from papers read before the American Institute of Mining Engineers.

The work concludes with an article on the Assaying of silver bullion, as practised by me when Assistant-Assayer in the United States Mint of San Francisco, and when in charge of mines and metallurgical works during a period of eighteen years.

17, Belsize Crescent,
South Hampstead, London, N.W.
April, 1889.

AUTHOR'S NOTE TO SECOND EDITION.

I am much gratified by the speedy sale of a large impression of the First Edition of this work; and I have taken the opportunity afforded by a Second Edition to add a Supplementary Chapter.

Johannesburg, South Africa.
June, 1891.

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I.—SILVER MINES OF THE WESTERN STATES OF AMERICA.

ON many occasions, when acting (as I did for a period of eighteen years) in charge of mines and metallurgical works in the Western States of America, it fell to my duty to visit professionally new mining districts, as they were opened up to civilization by intrepid prospectors, who had penetrated into unexplored mining fastnesses, inhabited only by wild Indians. To these pioneers—the advance guard of an ever-changing and shifting Western mining population—we owe numerous discoveries; and the world at large is indebted to them for a large amount of the stock of precious metal now available.

A warm tribute is due to the prospector for the services he has thus rendered, and is still rendering, to his fellow-men. Hardened by prolonged outdoor life they are noted for their indomitable courage and bravery, and their readiness to face hardships and privations of all kinds; and although their outward appearance, like their manners and diction, may be rough and uncouth, I have discovered among them traits of a noble character worthy of men in the highest ranks of life.

A party of these pioneers, who were located in 1859 in one of the mountain ranges east of the Sierra Nevadas, on what they supposed to be gold "placer" mines, while washing the auriferous earth found intermingled with their scales of gold dust, a black earthy substance which so seriously interfered with their method of gold extraction that it was decided that one of their party of five should go to Sacramento to ascertain what this black substance was. Examination of the "black stuff" revealed it to be pure sulphide of silver; and these five prospectors, it appeared, were "ground-sluicing" on the much-decomposed outcrop of the Ophir mine, a portion of the now famous Comstock lode, which was subsequently so named after Comstock, the leader of the party. This was the first discovery of silver in the United States of America; and it is hardly necessary to say that it created almost unbounded excitement among the miners of California. Shortly afterward the opening of the Esmeraldas, farther south in Nevada, led to the production of large amounts of silver; and these same mines are now being reopened by some London mining companies.

At the outset of metallurgical operations difficulties were experienced in the treatment of the ores, but in spite of all drawbacks, the quantity of gold and silver produced from the Comstock, within a short time after the discovery, was enormous, demonstrating the great wealth which was hidden in hose mines, which are productive even now.

Thanks to the efforts of some metallurgists with European raining and experience (amongst whom special notice is deserved by the Hungarian engineer, Guido Küstel), the treatment of the ores gradually improved, and it eventually de-

veloped into the now perfect Washoe Process, named after the district in which the Comstock Lode is located.

The next discovery of silver of any note was the deposit on War Eagle Mountain, in Idaho Territory; and in spite of the distance of the spot from any centre of supply—located, as it was, in an inhospitable region, difficult of access on account of roving Indians—a most prosperous mining district quickly sprang up. As one of the early settlers there, I became interested in some valuable finds; and I was probably one of the youngest of the mine and mill-owners in that region, having then barely passed my twenty-second year.

Most noted among the mines of the district were the Golden Chariot and Ida Elmore. Their working shafts were barely one hundred feet apart; but the short intervening piece of ground contained such riches that both mines coveted its possession, and came into frequent collision as to the exact boundary line. Their disputes culminated in armed contests deep underground, men armed with repeating rifles being sent down the shafts to protect the miners from behind breastworks built up in the tunnels and drifts, while the diggers were blasting out the rich ores from the disputed bonanza. As the working galleries of the mines were connected, several men were killed and wounded on both sides before this unsatisfactory state of things was brought to an end. It was costly work, for the companies had to pay their "fighters" from £100 to £200 a day for doing patrol duty in the mines.

Matters were brought to a crisis by the killing of Mr. Marion Moore, the largest shareholder of the Ida Elmore; and the country becoming terrorized, the governor of the territory came upon the scene with a battalion of United States soldiers, and taking possession of the mines, proclaimed martial law and restored order. That was in March, 1868. This was not the only instance of a bloody conflict for the possession of rich ore, similar scenes having been enacted in other districts.

Now followed discovery after discovery, and news reached us from the sandy deserts of Nevada that rich silver ore had been found in the Reese river regions. From that point a

party of prospectors started in 1869 for the Burnt Wagon District, seventy miles north-west, a country which took its name from the remnants found there of burnt wagons, which had belonged to a party of emigrants attacked and killed by Indians, the latter finishing their work by burning the wagons. Near by was a spring, and here the party of prospectors camped. Their search for minerals was rewarded by a most magnificent find of bold and rich outcroppings, which were located on what they designated "Mineral Hill." Their reward was the sum of £100,000, which they received from the Californian purchasers. When these mines were sold to an English company, in 1870, I was appointed metallurgist of the works. My engagement at this concern gave me the opportunity to examine and study the Eureka Silver Lead Mines, sixty miles farther south and thirty miles beyond the then celebrated White Pine Mines; and, to crown the brilliant silver period, the famous Pioche Mines were "located" by some soldiers, while in pursuit of hostile Indians, another hundred miles farther on in the south-eastern corner of Nevada. Meanwhile, and simultaneously, the western escarped ridges of the bold Wahsateh ranges were being closely searched. There large bodies of silver lead ore were uncovered, rivalling the Eureka Mines in extent. Such mines as the Emma, Flagstaff, and many others were also being developed, and the completion of the great Central and Union Pacific Railways, which crossed the continent, assisted the early explorers in the active prosecution of their enterprises.

Meanwhile, also, explorations were pushed farther south, and in 1874 the discoveries at Panamint, in Suprise Cañon, Inyo County, attracted me to that region. While there I had occasion to investigate the New Coso Silver Lead Mines, the Argus Range with its noted Modoc Mines, Wild Rose Springs and adjacent territories, including Death Valley, some portions of which are 110 feet below sea-level.

When I look back to those days, when I underwent privations and hardships such as were not known to me in a more congenial northern clime, I cannot but commiserate those poor

fellows who, falling a prey to hunger and thirst, lost their lives in their struggles to acquire wealth in those mountain fastnesses, and whose bones are bleaching among the desert sands of Panamint and Death Valley. Beyond California and Nevada, still farther south, among the more dreary and desolate reaches of Arizona, the indomitable energy and perseverance of the prospector opened up new countries and fields for enterprise. Arizona and New Mexico, although very sparsely settled and little known as far back as 1870, have since risen to the rank of great silver-producing countries, and the Silver King, the Tombstone Mines, and others too numerous to mention, have contributed their quota to the world's stores of silver and gold.

In the course of my explorations, I visited over twelve hundred mines and every individual district possessed distinct geological features of its own, and in no case did I observe two distinct districts, which, as to formation and character of ores, were entirely similar in structure. No amount of geological knowledge alone, therefore, will enable an observer to determine, as it were by rule of thumb, the possibilities or probabilities of a newly discovered mine; and I accordingly recommend the mining engineer or metallurgist who is in charge of a mine to thoroughly explore his ground and open up his ore bodies before launching into large expenditure in the erection of reduction works. Let him first satisfy himself that he has "plenty of grist for his mill."

Silver on the Pacific Coast of America.—As the demonstrations given in this work mostly refer to mining regions located on the Pacific coast, I will here quote an interesting account given by Mr. Clarence King* of the geological features of the mining districts and the mode of occurrence of the precious metal ores :—

"West of the hundredth meridian, and bordering the Pacific Ocean from Southern Mexico to the Arctic Sea, stretches a series of mountain chains and elevated plateaux. From a width of 400 miles on the Mexican line it rapidly widens to

* In his "Geological Explorations of the Fortieth Parallel."

the north, reaching its greatest expansion on the fortieth parallel, where the actual mountain zone is over 1,000 miles wide. From this point it again narrows toward the north, diminishing to 400 miles upon the north-west boundary line.

"The central portion of this area is embraced under the general name of the Great Basin, and includes all that barren middle ground between the California mountains and the Rocky system.

"The Great Basin is walled up upon the west by the Sierra Nevada range. Its eastern boundary is the Wahsatch chain. A section across the 500 miles of intervening country, along the fortieth parallel, shows a rapid descent from the Sierra summit, whose elevation at this point is about 10,000 ft., to a depressed zone of country which skirts the great range for 1,000 miles and spreads out to the east in comparatively level deserts, its surface here and there interrupted by abrupt chains of mountains. From this area of desert lowland, averaging in altitude about 4,000 ft. above sea-level, the country gradually ascends to the eastward, the surface being occupied by a succession of meridional ranges separated by trough-like valleys.

"Where the fortieth parallel intersects the one hundred and sixteenth meridian of west longitude the rise culminates, and passing thence eastward the general profile of the plateau sinks gradually to a second belt of depressed plains, whose aridity rivals those of the Nevada Basin. The desert basin of Utah is still further suggestive of Nevada since the lofty Wahsatch rises abruptly from its level, scored by sharp cañons closely resembling the eroded front of the Sierra. The average altitude of the entire system of parallel ranges, which trace themselves from north to south across the Great Basin, is not far from 9,000 feet.

"The plains, from a level of 4,000 feet, which is about the height of the basins of Nevada and Utah, rise to a mean elevation of 6,000 feet in the middle of the system.

"If a circle be traced from this central point, latitude 40 and longitude 116, with a radius of 250 miles, the circumference will be found to mark a continuous chain of depressions. The

middle plateau of Nevada is, then, an important centre of elevation and of drainage.

"The depressions of the ring are dreary reaches of desert where fields of sand alternate with alkaline plains, where, in the brilliant dry air, the eye may range over expanses of desolate lowland, naked and devoid of all vegetation except those blighted-looking forms of life, the sages, which rather intensify than relieve the deathly aspect of the scene. The central plateau region is somewhat more favoured; occasional grassy valleys interrupt the sage plains, and the mountains are less sterile and forbidding. Trees, which were almost wholly absent in the lowland, occur on the loftier ranges, and near the perpetual snow which here and there caps the higher summits an extraordinary fertility is developed.

"A glance at any physical map of this region shows a general parallelism of ridges with a prevailing north-west trend. The materials of this immense mountain area are infinitely varied, ranging from the earliest to the most recent deposits, and embracing almost all known species of sedimentary and eruptive products. The greater part of the rock is a series of conformably stratified beds, reaching from the early Azoic up to the late Jurassic period, when these level beds were compressed into vast mountain corrugations and elevated above the sea in a general wide and high plateau. Accompanying the upheaval and crumpling of this great oceanic family, and bursting from its fractured folds, are important masses of granite, penetrating the axis of the flexures and breaking through lateral fissures. Quartz-porphyries, felsite, and, notably, syenitic granite, with occasional occurrence of granulite and greisen, accompany the ejections of granite.

"The date of this orographical period is assigned to the late Jurassic; and Professor J. D. Whitney pointed out that the Sierra Nevada was folded after the Lias and prior to the Cretaceous, whose strata of sand and clay rocks repose unconformably upon the upturned and metamorphosed mass of Jurassic slates.

"This exploration has demonstrated that all the parallel

ranges of the Great Basin, including the chain of the Wahsatch, its eastern wall, belong to the same system of upheaval, and that, while the Pacific built upon the western base of the Sierras those fringing deposits of sand and clay which thickened through the undisturbed period of the Cretaceous and a wide range of the Tertiary, the Atlantic, or, more exactly, that ocean which covered the Mississippi Basin, beat upon the east flank of the Wahsatch and laid down a series of Cretaceous and Tertiary strata exactly corresponding with the coast deposits of the Pacific.

"At length, after accumulating to an extraordinary thickness, these outlying and later shore beds, subsequently to the Miocene, were themselves folded into mountains parallel and outside of the earlier system.

"As granites accompanied the upheaval of the earlier stratified group, so volcanic rocks have poured out from ruptures of the second mountain uplift. From the crests of the ranges, from the fissured bottom of the Pacific Ocean, from innumerable vents over the whole area of the western mountain system, there burst forth a series of volcanic eruptions which in many instances have overflowed and completely masked the earlier ranges, and in others have filled old depressions, building everywhere immense piles of lava mountains, and lifting here and there volcanic cones of the most impressive order.

"Long prior to the deposition of the great Paleozoic beds, a limited group of chains, composed of granites with crystalline schists and interstratified layers of specular iron, was lifted in Arizona, and probably over a considerable area of North Central Mexico. Later than the main Tertiary mountain building, which resulted in the Pacific coast ranges and the important chains east of the Wahsatch, a final series of disturbances has taken place. But neither are the earliest nor latest dynamical epochs of considerable geographical importance.

"A brief study of the map of the states and territories of the Pacific coast will teach the one great and prominent law of arrangement of Cordillera mountain chains, namely, that they trend from north to south, or north-west to south-east—in other

words, longitudinally with the main axis of the whole system of elevation. In strict subordination to this longitudinal direction of ranges are grouped all the structural features of local geology. The average strike of the great area of upturned strata is generally with the meridian. All the larger outbursts of granitic rocks conform to it as well, since their vents are most commonly the axial lines of actual folds; and, lastly, when the Tertiary uplift occurred, its ranges bordered the older mountains in parallelism, and the volumes of lavas accompanying it found exit through longitudinal vents, and either built themselves up along the ancient lines of structure, or, through new fissures, piled up chains of volcanoes conforming in trend with the general north and south plan.

"Over the whole Cordilleras are found localities of the precious metals; and it is not surprising to observe that, following its leading structural idea, they appear to arrange themselves in parallel, longitudinal zones.

"The Pacific coast ranges, upon the west, carry quicksilver, tin, and chromic iron. The next belt is that of the Sierra Nevada and Oregon cascades, which, upon their west slope, bear two zones, a foot-hill chain of copper mines, and a middle line of gold deposits. These gold veins and the resultant placer mines, extend far into Alaska, characterised by the occurrence of gold in quartz, by a small amount of that metal which is entangled in iron sulphurets, and by occupying splits in the upturned metamorphic strata of the Jurassic age. Lying to the east of this zone, along the east base of the Sierras, and stretching southward into Mexico, is a chain of silver mines, containing comparatively little base metal, and frequently included in volcanic rocks.

"Through middle Mexico, Arizona, middle Nevada, and central Idaho, is another line of silver mines, mineralised with complicated association of the base metals, and more often occurring in older rocks. Through New Mexico, Utah, and Western Montana lies another zone of argentiferous galena lodes. To the east again, the New Mexico, Colarado, Wyoming, Dakota, and Montana gold belt is an extremely well

defined and continuous chain of deposits, some of their ranges containing large deposits of lead and copper ores.

"In the history of the entire Cordillera there are, then, two periods of orographical disturbance, which have been accompanied by the rending of mountain chains and the ejection of igneous rocks. Such periods as these, of course, afford the conditions of solfataric action and the consequent formation of metal-bearing lodes. That period which culminated in the Jurassic produced over the entire system a most profound disturbance, and is in all probability the dating point of a large class of lodes. To the second or Tertiary period may be definitely assigned those mineral veins which traverse the early volcanic rocks.

"These two periods have produced two types of metalliferous lodes. First, those veins which are wholly enclosed in the granites, or in the more or less metamorphosed strata of that series which extends, with perfect conformity, from the Azoic up to the Jurassic. The latter are generally found occupying planes of stratification or jointings developed by metamorphism; and although they closely conform in dip and strike to the country rock, the clay selvages and striated surfaces of the quartz, together with a usually unbroken continuity downward, seem to indicate an origin similar to true fissure veins. Of this type are prominently the gold veins of California.

"The districts embracing what are known as the Humboldt mines are located upon two parallel ranges, formed almost wholly of folded strata of the Triassic age. The veins either occupy planes of stratification or arrange themselves along prominent jointing planes, induced by the disturbances of upheaval and metamorphism.

"The districts of Reese river occur first in a large mass of granite, accompanying a mountain fold, and secondly are found lying in the metamorphic rocks of the carboniferous, in position similar to those of Humboldt. The once celebrated White Pine District, whose mineral deposits are enclosed conformably between strata of Devonian limestone, is a prominent example of the groups comprised wholly within the ancient rocks.

"The discovery of the geological horizon of these limestones

by Professor Meek is of interest, since it proves them to be among the oldest known silver-bearing rocks in either of the Americas. The mode of occurrence and distribution of the mining districts of Colorado are somewhat unique. In general they belong to the ancient type.

"The second type belongs to the second or Tertiary oro graphical period, and finds its origin in the disturbances of the volcanic ejection. While the fires of the lava period were still burning, and where the deeply riven rocks of the earliest volcanic outflows were repeatedly broken through by subsequent eruptions, and where torrents of water poured down the hill slopes and everywhere penetrated the fissured rocks and came in contact with intensely heated material, there were present all the elements of vein formation. That these conditions actually existed is a matter of every-day geological proof. The lodes of this type are either wholly or in part enclosed in volcanic rocks.

"Many important veins of Mexico, several of those which border upon the Colorado River within the United States, and, in general, that zone which lies along the eastern base of the Sierra Nevada, are members of this family, as is clearly proven by the fact that they are either wholly or in part cased by volcanic rocks. The most prominent example of this type within the limits of this exploration is the Washoe district, whose remarkable Comstock lode, although in one place indistinctly touching the ancient formations, yet, as deep workings have shown, is chiefly inclosed by a modern volcanic rock, and evidently owes its origin to the later disturbances. The Owyhee district, in Central Idaho, occurs upon the crest of a granite mountain chain, which has been intersected by a series of volcanic dykes, ranging from the earliest propylite to the most recent basalt. From the peculiar association of the mineral veins with these dikes, and the manner in which they intersect each other, it is obvious that the quartz lodes belong to the Tertiary period.

"From these few but important general facts it will be seen that by far the greater number of metalliferous lodes occur

either in the stratified metamorphic rocks or in those ancient eruptive rocks which date from the period of Jurassic upheaval; yet very important, and perhaps more wonderfully productive, have been those silver lodes which lie wholly in the recent volcanic formations. It is evident from a careful study of the ranges that much of the dislocation and general mountain disturbance was occasioned by the Tertiary upheavals. How far the veins which lie wholly in the older rocks belong to this second disturbance it is impossible to say. In some instances it is evident that the veins themselves have been twisted and disturbed together with the strata; in others it seems most pro bable that the whole fissure and solfatara were induced by the latest movement. In the present state of knowledge it is impossible accurately to classify and catalogue the age of the fissures themselves. In almost all cases the evidence tends to the belief that the veins belong to the Jurassic period; and yet it should be borne in mind that wherever the more recent strata have been formed from the detrital materials of the older, we look in vain for the ore-bearing pebbles. The writer is not aware that even in the broad Tertiaries which fringe the west base of the Sierra Nevada any auriferous pebbles have been found; yet, at the same time, the Jurassic origin of these veins is proven from other data which were furnished by Professor Whitney. The fact, then, of the absence of the ore-bearing pebbles from the later strata is only negative evidence, and cannot weigh greatly against the probable Jurassic age of very many mineral deposits. So far as we now know, no metallic veins occur in the sedimentary formations of the Tertiary period east of the Wahsatch Mountains; whereas the remarkable metamorphism which has occurred during that period in the coast ranges, near San Francisco Bay, has developed extraordinary deposits of quicksilver and chromic iron.

"The metallic minerals have, then, in obedience to the prevailing rule of longitudinal structure, arranged themselves in parallel zones, which extend from north to south over great areas. In the history of the Cordilleras two prominent epochs of mountain building have taken place, each accompanied by

the ejection of igneous rocks with metamorphism and solfataras, and, lastly, with the formation of great numbers of metal-bearing lodes. The lodes belonging to the Jurassic age occur chiefly inclosed between strata ranging from that date down into the Devonian along the planes of deposition or the jointing planes developed by metamorphism and pressure; or lastly, in the igneous rocks of the granite family which accompanied the Jurassic disturbance.

"The veins of the second type, which belong to the Tertiary period, are found inclosed either in part or in whole within the volcanic rocks. Their position in that family is among the earlier members, never, so far as is now known, either penetrating the trachyte or basalt. While the greater number of veins probably belong to the first type, some of the most brilliant examples of metal lodes have occurred as members of the second type."

II.—Silver Ores.

Silver Ores.—Silver is not only found in its native (or metallic) state, but also as an amalgam and as an alloy, and in combination with various non-metallic elements—such as chlorine, bromine, iodine, sulphur, and selenium; it has also been detected in sea water.

Native Silver occurs crystallized in cubes, octahedra, and other forms related to these; sometimes it exists in laminated, filamentous, or amorphous masses, and sometimes very minutely disseminated through the minerals. It is found to accompany all its ores, especially the sulphides and chlorides. It is met with chiefly in the primitive formations, as in granite and gneiss; more rarely in the argillaceous schists and grauwacke of the transition rocks, accompanied by quartz, carbonate and fluoride of calcium, sulphate of baryta, carbonate of iron, galena, &c.

Nearly all the localities in which silver is known to exist have produced native silver, although frequently in very small quantities. It exists in the ferruginous rocks of Brittany, and

has been found at Kongsberg, in Norway, and in the Saxon mines at Freiberg, as well as in Mexico, Chili, Peru, Nevada, and—associated with native copper—in the district south of Lake Superior. The native metal is always found associated with copper, iron, or gold.

In some of these localities it has been found occasionally in considerable masses. Thus at Kongsberg pieces have been extracted weighing from 50 lbs. to 600 lbs.; in America, at the end of the last century, lumps of 200 to 800 lbs. were obtained; and on one occasion a block of solid silver was discovered in the mine of Johanngeorgenstadt, which is said to have weighed 9,000 or 10,000 lbs.

Vitreous Sulphide of Silver, or Argentite, contains 86 per cent. of metallic silver, and its chemical formula is Ag_2S. It is of a dark lead grey colour, may be cut with a knife, and shows a metallic lustre where cut. It crystallizes in cubes and dodecahedras, but generally occurs massive. It has a specific gravity of about 7·2, and is easily fusible, giving off vapours of sulphur, and leaving a button of metallic silver. It is met with in nature almost always combined with other sulphides, as those of copper and lead. This mineral is one of the richest and most abundant ores of silver; it forms a large proportion of that annually produced by the various continental mines, as those of Saxony, Bohemia, Hungary, and is particularly abundant in the mines of Guanaxuato and Zacatecas, in Mexico, and in conjunction with stephanite, native gold, &c., in Nevada.

Dark Red Silver Ore, Ruby Silver, or Pyrargyrite.—This mineral crystallizes on the rhombohedral system, and has a conchoidal fracture. Its specific gravity is about 5·8, and according to Wöhler it has the following composition:—

Silver	60·02
Antimony	21·80
Sulphur	18·00
	99·82

—a result which would indicate its formula to be Ag_3SbS_3. It is found, together with calcite, native arsenic, and galena, at Andreasberg, in the Hartz; also in Saxony, Norway, Hungary, and Spain, in the Old World; and in Nevada, Idaho, Mexico, and Chili.

Light Red Silver Ore, or Proustite.—Is a sulphide of silver and sulphide of arsenic, containing 65·4 per cent. of silver; sometimes the sulphide of arsenic is replaced by sulphide of antimony. It is found in Saxony and in Bohemia, also at Guadalcanal, in Spain, and in several localities in Nevada.

Brittle Sulphide of Silver, or Stephanite.—Crystallizes on the orthorhombic system, is a compound of sulphide of silver with the sulphide of antimony and arsenic, and has a nearly black iron-grey colour, a metallic lustre, and conchoidal fracture. It is generally found associated with other ores, both in Europe and in the various parts of the New World. According to the investigations of Rose and Klaproth, its composition is as follows:—

Silver	66·50
Copper and Arsenic	0·50
Iron	5·00
Antimony	10·00
Sulphur	12·00
Loss	6·00
Total	100·00

From which its formula would be Ag_5SbS_4, or $5Ag_2S+Sb_2S_3$.

Polybasite is another form of the brittle sulphide, which contains a small quantity of copper. Its crystals are short tubular prisms, and it is also found massive and disseminated. It has a semi-metallic lustre and an iron-grey colour, and the scales are blood red by transmitted light. Its specific gravity is about 6·2. It is distributed much as the last-named ore, and according to Rose the ores from Mexico and Schemnitz have the following composition:—

Silver	64·29
Antimony	5·09
Arsenic	3·74
Copper	9·93
Iron	0·06
Sulphur	17·04
	100·15

And its formula is $(AgCu)_9(Sb,As)S_6$.

Dark White Silver Ore (*Brittle Silver Ore*) is a combination of sulphide of silver with sulphide of antimony, with 18 to 31 per cent. silver and 15 to 26 per cent. of copper, and carrying also sulphide of iron and zinc in combination.

Light White Silver Ore contains 38 per cent. lead, 5·7 per cent. silver, and traces of copper, with sulphide of iron, zinc, and antimony.

Monochloride of Silver (or *Horn Silver*) is semi-ductile, and sufficiently soft to be cut with a knife; colour, pearl grey inclining to blue, and becoming brown in the air; has a vitreous lustre, is usually translucid, crystallizes in cubes. When pure it consists of silver, 75·3; chlorine, 24·7; and its composition is therefore represented by the formula AgCl.

Iodyrite, or Iodide of Silver, is a rare mineral of a pale lemon-yellow colour, with sometimes a tint of green. It is of rare occurrence. It is composed of silver, 77·4; iodine, 22·6.

Bromyrite, or Bromide of Silver.—Found in Chili and Mexico, also in Nevada, and on account of its green colour is called *Plata Verde* by the Mexicans. It is composed of silver, 57·70; bromine, 12·50.

Native Amalgam has a very bright silver-white colour, and is so soft as to be easily cut with a knife. It occurs both in distinct crystals, in irregular amorphous masses, and at times as thin compressed plates. It crystallizes in the regular octahedron

or dodecahedron. This mineral is found in a great many different localities, but the finest specimens have been procured from Moschellandsberg, in Bavaria. Its specific gravity is 14·1, and it contains 36 per cent. of silver and 64 of mercury, which would indicate its formula to be AgHg.

Argentiferous Galena.—The sulphide of lead, or galena, is almost always associated with a small quantity of silver in the state of sulphide.

Telluric Silver (or *Hessite*).—Containing 61 per cent. of silver, sometimes also gold, and traces of iron. It occurs in Siberia in a talcose rock, with pyrites and blende; specimens have been obtained a cubic foot in size.

Sternbergite.—A sulphide of silver and iron, containing 33 per cent. of silver. It is a highly foliated ore resembling graphite, and like it leaving a tracing on paper. The thin laminæ are flexible and may be smoothed out by the nail. Lustre, metallic; colour, pinchbeck brown.

Eucairite is a seleniferous ore of silver and copper occurring in black metallic films. It occurs in Sweden. Selenium, when found in Swedish mines, is the indicator for silver. It is also found in the Hartz mines. Before the blow-pipe fumes of selenium are given off, having an odour like decayed horse-radish.

Geographical Distribution and Occurrence of Silver.—The metal is found in rocks of various geological ages—in gneiss, granites, and allied rocks; in porphyry, trap, sandstone, limestone, and shales; and the sandstone and shales may be as recent as the middle secondary. From the variety of the ores enumerated above, it will have been seen that they are associated often with other ores, such as lead, zinc, copper, cobalt, and antimony—with which metals they also enter into combination. The usual gangue of silver ores is quartz or calc spar, but they are also found in fluor spar and heavy spar.

The ores in Mexico are nearly identical with those found in South America, consisting of horn, or chloride of silver, the antimonial varieties, the black sulphides, and native silver.

Mexico has always been noted as a great silver country; the western slope of the Cordillera, especially, is furrowed with many silver veins, and the principal districts are Zacatecas, Guanaxuato, Fresnillo, Sombrerete, Catorce, Oaxaca, Pachuca, Real del Monte, and Pasco. Some of the richest silver veins in Mexico occur in chloritic shale and porphyry. At Zacatecas the veins are in graywacke. Some prolific deposits also occur in limestone.

The Chilian mines are also on the western slope of the great Cordillera, and occur where the shales and sandstones are intersected with porphyry. The ores are decomposed on top, but at a depth of 300 ft. to 400 ft. they change into the less tractable antimonial and arsenical combinations.

Peru has a brilliant example in the Cerro de Pasco mine, at an altitude of 15,700 ft. in the Andes. Back of Iquique are the Huantaya mines. In the Argentines are the noted Potosi mines.

In Europe, half-a-dozen countries—Spain, Sweden, Norway, Saxony, Austria, and Russia—furnish their contingent of this precious metal; and in England argentiferous galena is worked for its silver.

Chemical Properties of Silver.—At ordinary temperatures silver is not acted upon by dry or moist air; but it is tarnished if exposed to an atmosphere containing very minute portions of sulphuretted hydrogen, which is always present, to a greater or less extent, in the air of chambers heated by coal fires. When melted in open vessels it possesses the remarkable property of absorbing mechanically about twenty-two times its own bulk of oxygen, which it disengages on solidifying, producing the metallic vegetation observed on the surface of a silver button when suddenly cooled in a cupel. If a mass of pure silver, after being maintained for some time exposed to

the air in a state of fusion, be allowed to cool suddenly, the outer coating becomes solid, and this crust becomes soon broken, partly by the liberation of the oxygen which has been absorbed, and partly, perhaps, by the expansion which is said to take place when silver solidifies. This phenomenon, which is called *rochage* or *spitting*, occasions the loss of a small quantity of silver from the mass. It does not occur when the silver is alloyed with a small proportion of copper, gold, or lead; 1 per cent. of copper prevents the absorption of oxygen, to which the phenomenon is due.

If subjected to a very high temperature, such as that produced by the oxyhydrogen blowpipe, silver is rapidly volatilized.

Silver does not absorb the oxygen of the air when it is fused with the alkalies; and for this reason silver crucibles are used in the laboratory for making analyses of silicates by means of caustic alkalies, by which platinum would be attacked. Chlorine, bromine, and iodine combine very readily with silver, and of these three bodies iodine has the greatest affinity for it. Of the acids, nitric acts most powerfully on it, dissolving it in the cold; concentrated sulphuric acid requires heat to dissolve it. Hydrochloric acid attacks it only with difficulty, by leaving it a long time to digest; however, if the metal is in a very divided state, and the liquid be heated to ebullition, the solvent action is considerable. Silver combines readily with sulphur.

III.—Treatment of the Ore.

Metallurgy of Silver.—The separation of silver from its ores is effected in various ways, according to their nature.

Of the two methods which will be considered in this volume, the first consists in forming an amalgam of silvery mercury, from which the mercury is expelled by the aid of heat.

In the second, the metal is reduced to either a chloride or a sulphate and extracted by lixiviation, and then precipitated from the solution.

There are also processes of a different character by which alloys of lead and silver are formed by methods dependent on the nature of the ore, but these are reserved for consideration in a separate volume.

Special treatment is also required for argentiferous copper ores. which will be considered in a separate volume.

The metallurgy of silver has been brought to its present state of perfection through the discoveries of the silver mines in Nevada over twenty-eight years ago, to which I have referred at the outset of this chapter. There the conditions were such as to call for special treatment, which led to many improvements in the machinery previously employed, and brought the whole question of silver-mining into prominence. In Central America methods of working silver ores are still practised which can only be used in the localities as local conditions and climate permit, but which in some instances cannot be superseded by any other method, although to the skilled metallurgist they may seem out of date. Still, in spite of the crudeness of the methods, the ores are worked as close and perfect as in any of the large silver mills of Nevada, and these crude systems have laid the foundation of the present silver metallurgy, which has added so many millions to the world's stock of precious metals.

A brief account of these systems may therefore be given here.

Mexican or Patio Process.—This method was invented by Bartholome Medina, a native of the town of Pachuca, in Mexico, in 1557. It is still practised in that country in all its primitive simplicity; and perhaps when the cheapness of labour, the poverty of the ores, the scarcity of fuel, and the deficiency even of water-power are considered, it could not be profitably superseded by any other. It will be found that its principal characteristics depend on these circumstances.

The ore, on being extracted from the mine, is placed in the hands of the *pepenadores*, men and women who break all the larger pieces with hammers; and after rejecting those in which

their experience teaches them that no metallic particles are contained, and setting aside those which are very rich to be treated by the smelting process, they subject the rest to a process of crushing and pulverization with a view to its direct treatment by amalgamation.

The ores destined for this treatment are reduced to lumps about the size of the fist, submitted to the action of the *ingenios*, or stamping mills, which are either driven by mules or, when water-power is at hand, by means of a small breast-wheel. The long horizontal shaft fixed on the axis of the wheel is armed with five or six cams, placed at different situations round the

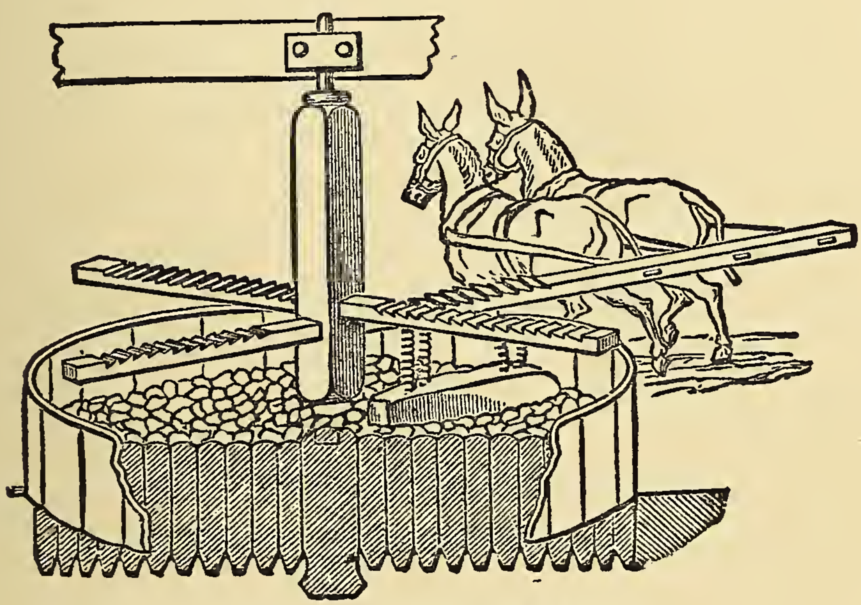


FIG. 1.—THE ARRASTRA.

shafts, so as to act in succession on the projecting teeth of the upright wooden lifters, shod with iron, each weighing about 200 lbs., and works in a corresponding mortar, in front of which is placed a screen perforated with small round holes. It is estimated that a battery of eight of these stampers is capable of reducing to powder two or three tons of ore in twenty-four hours.

The powder thus obtained not being sufficiently fine for the purpose of amalgamation, it is transferred from the stamps to be reduced to an impalpable slime or mud in the crushing mills, or *arrastras*, of the form represented in Fig. 1. These

mills are commonly worked by mules, which turn a vertical shaft armed with two cross bars. The grinding stones, as well as the sides and bottom of the mill (which are paved with great care, no interstices being allowed to exist between the pyramidal stones), are composed of porphyry, four blocks of which revolve in each crushing mill attached to the several arms.

The arrastras are usually about 12 ft. in diameter, and a number are arranged in a shed. The details of their construction will be at once understood from the engraving.

Gold is found in considerable quantities in conjunction with silver at Guanaxuato, in Mexico, and hence the grinding is conducted with more than ordinary care. From 600 to 1,100 lbs. avoirdupois of the ore, as delivered from the stamps, are introduced into each arrastra, and about 110 gallons of water gradually added during twenty-four hours; at the end of which period the *lama*, or argentiferous mud, is baled out and removed to reservoirs, where part of the water is evaporated by the sun's heat, and the mass is then ready for treatment in the *patio*. At Zacatecas no gold is contained in the ore; each apparatus is therefore capable of grinding nearly 2,000 lbs. avoirdupois in the course of twenty-four hours, since it is not necessary to reduce to such an extreme state of subdivision. A considerable quantity of mercury is lost during the process, probably in consequence of sulphide of silver being decomposed with the formation of sulphide of mercury.

The patio, or amalgamating floor, to which the lama is conveyed after it has been brought to a certain consistency through the evaporation of water by the sun's heat, is represented in Fig. 2. It is a large space, usually paved, to which a slight inclination is given in order to insure the removal of water, and in which are marked out a number of circular spaces, about 40 ft. in diameter, surrounded by low stone walls or frames of wood.

At Zacatecas the patio is rectangular, 312 ft. in length by 240 in breadth, and capable of containing 24 flat circular heaps of lama, each about 50 ft. in diameter and 7 in. deep,

arranged in four rows. These heaps are termed *tortas.* A small space is usually reserved at one corner for the purpose of performing assays on the ore, with the view of determining beforehand the proportion of mercury it may be necessary to incorporate with each heap.

A mass of lama weighing from 50 to 100 tons, according to the locality, is now introduced, and forms a layer about 1 ft. thick, which is allowed to further dry until of the consistency of moderately thin mud. From 3 to 5 per cent. of salt is added, and the first treading by mules takes place, after which

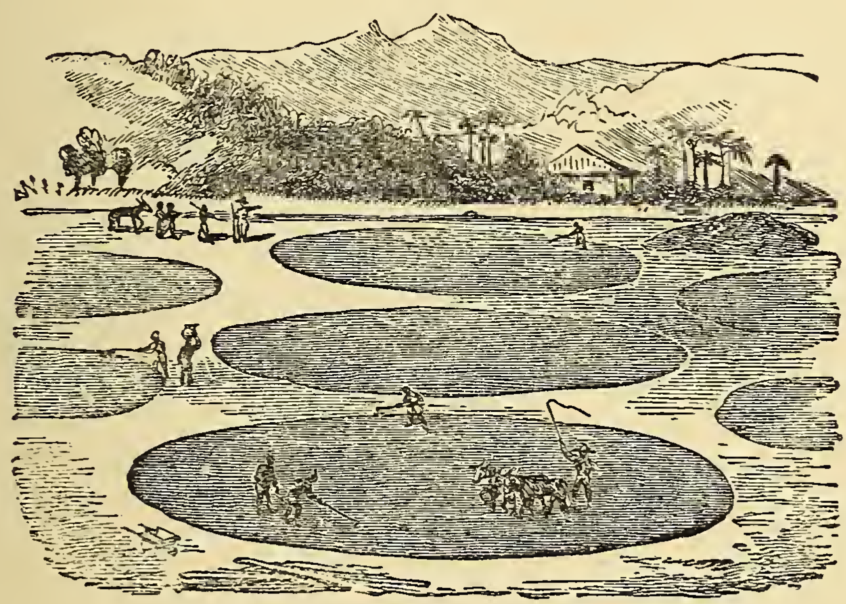


FIG. 2.—THE PATIO.

the mixture is allowed to rest until the following day, when mercury and magistral are added. This latter is prepared by roasting, in a reverberatory furnace, copper pyrites containing a considerable quantity of iron pyrites and a small quantity of salt. The sulphates of copper and iron, mixed with some chlorides, are formed, about 20 per cent. of the whole being sulphate of copper. Usually about 1 per cent. of magistral is added to each torta, but the exact amount varies according to the richness of the argentiferous ore and the season of the year. In summer, for example, and with the richer ores, about 750 lbs. to the torta, and in winter only half that quantity; for it is

stated as a singular fact that in summer the mixture cools and requires more warmth, while in winter it acquires of itself additional heat. The larger proportion is for minerals containing 0·0015 of metallic silver. When the operation proceeds too rapidly, arising from the presence of too much magistral (which would occasion a greater loss in mercury), the remedy is to add a certain quantity of lime, which serves to cool the lama.

In some localities the mass is trodden at this stage for about an hour, but it is more usual at once to distribute mercury over the surface in the proportion of about 1 lb. for every 1,000 grains of silver supposed to be present. This is effected by sprinkling the metal from a bag of coarse cloth, after which the mules are caused again to traverse it at intervals for several days.

The progress of the amalgamation is carefully determined from time to time by observing the aspect of the mercury obtained by washing the mineral in a bowl. During the first few days the mercury can be removed from the globule of amalgam obtained, which has a slightly grey colour, by simple pressure with the finger: and each day the amalgam obtained is more grey and more solid. If the change takes place slowly it indicates that probably an additional quantity of magistral is required; while, should the mercury become leaden, it shows that there is an excess of this substance, and the torta must be cooled by adding lime. Should the amalgam obtained be extremely hard, it will be necessary to add a small additional quantity of mercury, and sometimes even a third charge of this metal is required. In summer a torta may frequently be worked off in fifteen days, while as many as forty-five days may be required in winter.

After the treading is complete, it is at some mines usual to add a considerable quantity of mercury to the mass in order to collect the grains of amalgam.

The next operation is the washing of the amalgamated ore for the purpose of removing the earthy matters, and of thereby obtaining the amalgam or mixture of silver and mercury in a separate form. This is performed in *lavaderos*, or washing

vats, which are circular in form and solidly built in masonry, each about 5 ft. deep and 9 ft. in diameter.

At Guanaxuato the washing apparatus consists of three such tubs, which communicate with each other by openings at different elevations, and here no further addition of mercury is made; whereas at Zacatecas only one cistern is employed, a large quantity of mercury having been previously added. A horizontal toothed wheel, mounted on a shaft worked by mules, communicates, through the intervention of another toothed wheel, a movement of rotation to a vertical shaft placed in the middle of the vat, and armed at its lower part with four agitators consisting of cross-beams, from which rise long wooden teeth to the height of 5 ft. A stream of water continually flows into the vats from a tank on a higher level. Under the action of the agitators the lighter earthy matter is kept afloat, while the heavier amalgam sinks to the bottom, and from time to time the former is allowed to flow out into a second similar apparatus, where it is subjected to a second washing, and then allowed to run away. An entire torta of amalgamated ore may thus be passed through one vat in twelve hours.

At Zacatecas a considerable quantity of amalgam results from the system of washing adopted, and it therefore becomes necessary to pass the heavier residues, collected in the vats which receive the liquid from the tub, over a washing-table, after which they are roasted and returned to the arrastra. At Guanaxuato the agitators of the three vats are driven by the same shaft at slightly different velocities, by which the amalgam is more completely extracted. In certain districts these agitators are replaced by shovels, and the supernatant liquid is passed over riffle boxes.

A large quantity of silver and mercury is always lost in this process, the exact proportion depending on the nature of the ores, as well as the manner of treating them. At Guanaxuato the loss is, according to Phillips, from 9 to 14 per cent. of silver, at Fresnillo often 28 per cent., and at Zacatecas, according to Dupont, as much as 35 to 40 per cent. The loss of mer-

cury is about 12 or 16 ounces for every mark (3550 grains) of silver extracted.

The rationale of the process of patio amalgamation, as above described, may be explained as follows. In America the silver exists in the ores, partly in the native state, partly as a chloride, and partly as a simple or multiple sulphide. The copper sulphate which is added reacts on the chloride of sodium employed, producing sulphate of soda and bichloride of copper; the latter acts as an energetic chloridizing agent on the sulphide of silver in the ore, changing it into chloride of silver, and passing at the same time itself into the state of monochloride, which is dissolved by the solution of chloride of sodium, and in this condition reacts on the unreduced sulphide of silver, forming chloride of silver and sulphide of copper. The chloride of silver in the presence of a solution of salt is in its turn reduced by the mercury added, and by such arsenic, antimony, copper, lead, and tin as may be present, forming monochloride of mercury and an amalgam of silver. It is the monochloride of mercury so formed that is carried away by the water in the process of washing, and constitutes almost the whole of the loss in mercury. The chloride of sodium employed serves not only to transform the sulphate of copper into dichloride, but also to dissolve the chloride of silver, and thus to facilitate considerably its reduction by the mercury.

The Chemical Reactions of this process are expressed by the following formulas, according to Professor Egleston:—

$$CuSO_4 + 2NaCl = CuCl_2 + Na_2SO_4.$$

The chloride of copper acts on the metallic silver and the sulphide of silver; chlorides of silver are formed, which are dissolved in the excess of chloride of sodium.

$$2Ag + 2CuCl_2 = 2AgCl + Cu_2Cl_2.$$
$$Ag_2S + CuCl_2 = 2AgCl + CuS.$$

When mercury acts on artificially prepared chloride of silver it reduces it to a metallic state, when it enters into combination with the mercury.

$$2AgCl + 2Hg = HgCl_2 + Ag_2Hg.$$

This reaction takes some time, and is less sensible on the natural than on the artificial substance.

If chloride of copper is treated with mercury, sub-chloride of copper and sub-chloride of mercury are formed.

$$2CuCl_2 + 2Hg = Cu_2Cl_2 + Hg_2Cl_2.$$

This reaction takes place more rapidly than with chloride of silver. If chloride of iron is substituted for the chloride of copper, all the reactions take place, but much more slowly, and this is especially true when sulphide of silver is present. The presence of salt accelerates the reactions in all cases. If any of the metallic silver in the ore has not been transformed into chloride, this is attacked directly by the mercury.

When sulphide of silver and mercury are shaken together, sulphide of mercury and amalgam are formed.

$$Ag_2S + 2Hg = Ag_2Hg + HgS.$$

This reaction is slow, but much quicker than with chloride of silver. All the sulphide of mercury is lost.

The loss of mercury occasioned by its conversion into chloride is very considerable, and it might to a great extent be avoided by employing other metals, such as zinc, tin, lead, and copper. Thus the use of an amalgam of copper has given excellent results at Guadelupe y Calvo. The copper alone is dissolved by reducing the chloride of copper in the magistral to a minimum, and the mercury thus does not act until after the complete solution of the copper. The copper employed must be about one-third of the weight of silver contained in the ore.

The following analysis of the ore in the neighbourhood of Guanaxuato is given by Phillips :—

Silver	1·04
Iron	4·71
Copper	0·55
Sulphur	6·79
Carbonate of Lime	8·25
,, ,, Magnesia	3·26
Silica	75·00
	99·60

This specimen was of more than ordinary richness. An average sample of the ores worked by this process only contains about 0·15 per cent. of sulphide of silver, or 0·13 per cent. of the pure metal.

Theory of the Process.—Mr. Bowering denies that chloride of silver is formed at all, as none was found by him in a torta on the patio, during a constant examination extending over four months. In support of this theory, he says that when only two of the reagents — sulphide of silver, chloride of sodium, or sulphate of copper — are mixed together no effect is produced, and that when three are mixed in a small vessel, the mercury combines with just half of the chlorine in the chloride of copper and forms sub-chlorides of both metals. As the chloride of copper has the property of absorbing oxygen, he concludes that it is the principal reagent. According to this, mercury acting on the chloride of copper makes sub-chlorides of both. The chloride of copper absorbs oxygen, which acts on the sulphide of silver and makes sulphuric acid, and leaves the silver in a metallic state to be absorbed by the mercury. The sulphuric acid set free acts on the chloride of sodium, and forms sulphate of soda. Chlorine is given off, combines with the sub-chloride to make a chloride of copper, which is again decomposed, and so on. In this case the sub-chloride acts just as nitric acid does in the manufacture of sulphuric acid. The action of the chemicals in the pile is especially slow if sulphate of silver is present, in which case the loss of mercury is also very large. When the whole of the silver is in the state of sulphide, a large part of it, which may sometimes be as high as 40 per cent., is lost. The mercury transforms the chloride of copper into sub-chloride, which, like chloride of silver, is soluble in an excess of salt. The sub-chloride in this state acts more energetically on the sulphide of silver than the chloride. A sulphide of copper is formed while the silver is precipitated, and the chloride of copper formed again by giving up half the copper, which becomes a sulphide. This advantage is gained

only at the expense of a very large quantity of mercury; and in order to prevent this loss, experiments were made of not introducing the mercury until much later in the process, but this did not succeed, as the extraction of the silver was not so well done.

During a visit to Mexico several years ago, I myself had occasion to make the following observation. While exploring a mine near a small hacienda, I noticed that the proprietor, although he had a quantity of ore on hand, was not working, and on asking him the reason, he informed me that he had no magistral. One pile of ore was blue ore, or *colorados*, from the presence of copper carbonates, and after searching the village we procured at the drug store a few bottles of sulphuric acid. After reducing to torta a few tons of this ore, some sulphuric acid was added, and after several days the ammonia reaction showed the presence of the copper sulphate. I now added the salt, and working the torta again, quicksilver was added, and at the end of the washing the extraction of silver was quite satisfactory.

On trying the same experiment with the white ore showing rich specks of negras or argentite, the result was not satisfactory, and consequently the operation was finished satisfactorily by mixing the negras with the colorados.

Stove Amalgamation.—In certain parts of Mexico the process is modified by transferring the lama from the patio (which is under a shed), when the process is about half completed, to a stove consisting of a chamber with flues so arranged as to lead heat from a fireplace to the mixture. The heating is continued for about three days, and the mass is then returned to the shed for the completion of the patio process. The time employed is somewhat diminished and the yield of silver increased slightly, but the loss of mercury is excessive.

Hot Process of Amalgamation.—This process, much employed in South America, but to a very small extent in Mexico, is especially applicable to ores in which the precious metal exists as a halogen salt. The ore, after being

ground in the arrastra, but not to such an extreme degree of fineness as in the patio process, is transferred to a tub called a *cazo*, the sides of which slope, and are 18 in. high, and formed of wood or stone; while the base consists of a sheet of copper about 2 in. thick and 2 ft. in diameter. The whole rests above a hearth, by means of which the contents can be caused to boil. About 100 lbs. of ore are introduced, made into a paste by the addition of water, and heat is applied. From 5 to 10 per cent. of salt is now added, and the workman commences the addition of mercury, at the same time keeping the mass constantly agitated by a wooden stirrer. Mercury is gradually supplied until about twice the weight of silver the ore is assumed to contain is present, which point should be attained after the lapse of about six hours. The resulting slimes are subsequently treated by the ordinary patio amalgamation, without, however, the addition of magistral. In some districts the dimensions of the cazo are so far enlarged as to contain about 1,500 lbs. The operation on this large quantity is completed in the same time as when the smaller apparatus is employed. The loss of mercury in both of these cases is far less than in the ordinary way, amounting only to 2 or 3 per cent., and this is owing to the reduction of the chloride of silver at the expense of the copper bottom instead of by the mercury.

CHAPTER II.

THE WASHOE PROCESS, OR FREE MILLING ON THE COMSTOCK LODE.

CLASSIFICATION OF ORES—Separate Treatment—Crushing in Rock-breaker and Stamps—Grinding and Amalgamation in Pans—Use of "Chemicals"—Settlers and Separators—Straining the Amalgam—Re-torting and Melting—Tailings—Results of the Process—Loss of Quicksilver.

Classification of Ores.—The ores of the Comstock lode consist chiefly of various sulphuretted forms of silver, native silver, and gold—finely, almost imperceptibly, disseminated through a gangue of quartz. With these are associated a few other accessory minerals in inconsiderable proportions.

For metallurgical treatment the ores were in former years divided into two classes, according to their richness, the chief object of the classification being to separate those ores whose mineral composition—and, more especially, whose high value—demanded a very exact and careful treatment in order to obtain the highest possible percentage of their precious contents, from those of lower grade, which by reason of their inferior value had to be treated by less expensive methods.

About twenty-five to thirty per cent. of the whole value contained in these ores was gold, the remainder silver. In the bullion produced the relative proportion of the gold was a little higher, as it is more easily saved than the silver.

The silver of the first-class ores is intimately combined with sulphur, zinc, lead, iron, and other base metals, which render the extraction of the silver difficult. They could not be profitably treated by the simple methods applied to the more

docile ores of the second class, but were crushed dry, roasted with salt in reverberatory furnaces, and then amalgamated in barrels by what is known as the Freiberg process, which is still used at the present time in some European metallurgical works. Ores worth from £70 to £100 per ton were formerly treated by this method. Ores of the second class were treated by the method known as the pan process. The separate treatment of ores of the first class has now been entirely abandoned, and was only carried out in the early part of the history of the famous Comstock lode.

In this chapter will be given a detailed description of what is known as the "Washoe process" of amalgamation,* as applied to the former second-class ores, and as now carried out on ores of all grades on the Comstock lode.

Crushing.—The ore to be treated by the ordinary Washoe process is delivered from the mine to the mill in pieces varying in size from fine particles to those as large as a man can lift. It needs first to be crushed to a fine condition. This operation is performed by stamps—that is, heavy iron pestles lifted and allowed to drop in iron mortars into which ore has been thrown. The larger pieces of ore are first broken to a suitable size for feeding the stamps, either by a sledge hammer or a mechanical rock-breaker, Blake's machine being in general use for this purpose. From the rock-breaker the ore discharges into self-feeders, which feed the mortars automatically.

The stamps vary in weight from 750 to 950 lbs.; they are lifted and drop about 8 or 9 in., making from seventy to ninety blows per minute; they are arranged in batteries, which consist each of one mortar, with usually four or five stamps. Wet crushing is always employed for these ores; that is, a stream of water is admitted to the mortar with the ore, and, flowing off, carries with it the pulverized ore as soon as the latter is suffi-

* For portions of this description I am indebted to Mr. Clarence King's excellent reports to the United States Government or the "Mineral Resources and Mining Industry of the Pacific Coast."

ciently reduced in size to pass through the screens placed in front of the discharging apertures of the mortar.

The screens through which the crushed material is discharged from the mortar are either of brass wire cloth, having thirty or forty meshes to the lineal inch, or more frequently of Russia sheet iron, perforated with fine holes. Screens of the latter sort, in general use, are known as Nos. 5 or 6. In the last named the hole has a diameter of one-fortieth of an inch.

The stuff being discharged from the battery is conveyed in troughs, by means of the flowing water, to settling tanks, of which there is a series placed in front of the batteries. These tanks are usually built of plank, are 3 or 4 ft. deep by 5 or 6 or more ft. square, and are so arranged as to have communication with each other near the top, so that the stream of water, carrying the crushed ore in suspension, having filled one tank may pass into the next, and so on through several, depositing the material and not finally leaving the tanks until it has become tolerably clear. The number of tanks must be sufficient to allow of a certain portion being emptied while others are receiving their supply, and the conveying troughs are provided with gates so arranged that the stream can be admitted to one portion of the tanks and shut off from the other at pleasure. The stream, having deposited in these tanks the bulk of the material, is still charged with slimes, or rock reduced to an impalpably fine condition, which is only settled by a slow process. For this purpose the stream is sometimes permitted to pass through other large settling tanks, or to slowly deposit its charge in a pond or dam outside the mill, where such an arrangement is possible. These slimes form a variable and in some mills a large percentage of the whole amount crushed; in some instances, it is stated, more than 10 per cent. When one or more of the settling tanks in the mill have been filled the stream is diverted from such to others that have been emptied, and the full ones are in their turn cleaned out, the sand or crushed ore being then subjected to the grinding and amalgamating process of the pan.

Grinding and Amalgamation.—Pans.—The pans employed for this purpose present a great variety in the details of construction. Since the first "common pan"—a very simple form of apparatus—came into use, many inventors have exercised their ingenuity in devising improvements, and at present there are several different patterns, each of which has some special claim for excellence, and finds its advocates amongst mill-men of particular districts. A more detailed description of some of these will be found further on. The common features are a round tub (see *post*, Figs. 18 to 26), either of cast or sheet iron, but sometimes with wooden sides, 4 to 6 ft. in diameter and about 2 to 3 ft. deep, having a hollow pillar cast in the centre, within which is an upright shaft projecting above the top of the pillar that may be set in revolution by gearing below the pan. To the top of this shaft is attached, by means of a key or feather, a yoke or driver by which the muller or upper grinding surface is set in motion. To the bottom of the pan, on the inside, is fixed a false bottom of iron, cast either in sections, commonly called dies, or in one piece, having a diameter a little less than that of the pan, and with a hole in the centre adapted to the central pillar. This serves as the lower grinding surface.

The muller is usually a circular plate of iron, corresponding in size and form to the false bottom, and in diameter nearly equal to that of the pan, and a flat, conical, or conoidal form, according to the shape of the pan bottom. Its under side is furnished with shoes or facings of iron, about an inch thick, that may be removed when worn down and replaced by new. The muller is attached to the driver, which is put on and over the central pillar of the pan, and being connected with the interior upright shaft as above described is thus caused to revolve. There are various appliances for raising or lowering the muller so that it may rest with its whole weight upon the pan bottom in order to produce the greatest grinding effect, or be maintained at any required distance above it when less friction or more agitation is required. Various devices are also in use for giving proper motion to the pulp, so that when

the muller is in revolution the material may be kept constantly in circulation, passing between the grinding surfaces and coming into contact with quicksilver. Some pans are cast with a hollow chamber an inch or two deep, in the bottom, for the admission of steam in order to heat the pulp, while others employ only "live steam," which is delivered directly into the pulp by a pipe for that purpose.

The operation of the pan consists in the further reduction or grinding of the stamped rock to a fine pulp, and in the extraction of the precious metals by amalgamation with quicksilver. The quantity of ore with which a pan is charged for a single operation varies from 600 or 800 to 4,000 or 5,000 lbs., according to the size of the pan. The ordinary charge of pans, most generally in use at present, is 1,500 to 2,500 lbs.

In charging the pan the muller is raised a little from the bottom, so as to revolve freely at first. Water is supplied by a hose pipe, and at the same time the sand is thrown into the pan with a shovel. Steam is admitted, either to the steam-chamber in the bottom of the pan or directly into the pulp. In the former case the temperature can hardly be raised as high as in the latter; but, on the other hand, when steam is introduced directly, care is necessary to avoid reducing too much the consistency of the pulp by the water of condensation. The pulp should be sufficiently liquid to be kept in free circulation, but thick enough to carry in suspension, throughout its entire mass, the finely divided globules of quicksilver. In some mills both methods of heating are employed in the same pans the temperature being first raised with each charge by live steam, and afterwards sustained by admitting steam to the chamber only. Some pans are covered with wooden covers to assist in retaining the heat. When properly managed the temperature may be kept at or near 200° Fahrenheit. When in the use of live steam the pulp becomes too thin, the supply of steam is cut off, the covers removed, and the pulp allowed to thicken by the evaporation of the water. The steam in the chamber may keep the temperature up to the desired point in the meantime. Another advantage of the steam chamber is

that the exhaust steam from the engine may be used in it, while for use in the pulp it is better and customary to take steam directly from the boilers, because that which comes from the cylinder of the engine is charged with oil and is injurious to amalgamation. The muller is gradually lowered after the commencement of the grinding operation, and is allowed to make about 60 or 70 revolutions per minute. In the course of an hour or two the sand should be reduced to a fine pulpy condition. When this has been accomplished—and some mill-men prefer a still earlier stage (even the beginning) of the operation—a supply of quicksilver is introduced into the pan, the muller slightly raised from the bottom to avoid too great friction, which would act to the disadvantage of the quicksilver, and the action continued for two hours longer, during which the amalgamation is in progress. The quicksilver is supplied by pouring it directly over the charge in the pan, by trying to scatter it upon the pulp in a finely divided condition. The quantity varies greatly in different mills, the ordinary supply being about 60 of 70 lbs. to a charge of ore consisting of 1,200 or 1,500 lbs. In some mills a quantity, varying from 75 to 200 or even 300 lbs., is put into a pan when starting up after a clean up, and subsequently a regular addition of 50 or 60 lbs. made with each charge.

Use of " Chemicals."—To promote amalgamation it is the general custom to add to the charge, either at or soon after the beginning of grinding, or when supplying the quicksilver, various materials generally described as " chemicals," and now usually consisting of sulphate of copper and salt. Since the first introduction of the pan process a great variety of substances, supposed to effect the decomposition of the silver sulphurets and to facilitate amalgamation, have been suggested by process vendors, and employed by men possessing little or no knowledge of the science of chemistry. Even tobacco juice, decoction of sage bush, and various other equally absurd ingredients, are said to have been used by some operators and believed to be effective reagents in decomposition and the amalgamation

of the silver. The long list of materials once in use has now been reduced, excepting in a few places, to sulphate of copper and salt.

The quantity used varies from a quarter or half a pound to three or four pounds to each charge of ore, the two substances being employed in very variable proportions in different mills. Their action, however, which is generally supposed to be analogous to that produced by the same reagents in the Mexican patio process (see *ante*, p. 20), is but imperfectly understood, and their efficiency, at least in the manner and proportions in which they were formerly employed, may well be doubted. This is apparent from the fact that in some mills both sulphate of copper and salt were used; in others only the first was used without the second; and in others only the second without the first—or, if at all, in proportions so minute that its efficient action is incredible; others have dispensed with the use of "chemicals" altogether, and under all these varying circumstances equally good results have been obtained. Some mills, accustomed to use both salt and sulphate of copper, have dropped either one or the other while working continuously on the same kind of ore, without perceiving any difference in the result; and it is the opinion of many intelligent mill-men that neither salt nor sulphate of copper, in the manner and quantity as formerly employed, is essential to the efficient working of the ore in pans.

Two hours having been devoted to the grinding, and two or three more to amalgamation, the pan is discharged and its contents received by a settler or separator. The discharge of a pan is usually aided by a supply of water, which dilutes the pulp and permits it to run freely from the pan into the settler.

The pan, being emptied and partly washed out by the stream of water, is again charged with a fresh quantity of sand, and the grinding operation is resumed without delay.

Settlers or Separators.—These, like the pans, differ somewhat in details of construction, but they are usually round

tubs of iron or of wood with cast-iron bottoms, resembling the pans in general features, but larger in diameter.

The settler is usually placed directly in front of the pan and on a lower level, so that the pan is readily discharged into it. In some mills two pans are discharged into one settler, the operation of settling occupying four hours, or the time required by the pan to grind and amalgamate another charge. In other mills the settling is allowed only two hours, and the two pans connected with any one settler are discharged alternately.

The consistency of the pulp in the settler is considerably diluted by the water used in discharging the pan and by a further supply, which in many mills is kept up during the settling operation. In other mills, however, the pulp is brought from the pan into the settler with the addition of as little water as possible, and allowed to settle for a time by the gentle agitation of the slowly revolving muller, after which cold water is added in a constant stream. The quantity of water used, affecting the consistency of the pulp, and the speed of the stirring apparatus, are important matters in the operation of settling or separating. Since the object of the process is to allow the quicksilver and amalgam to separate themselves from the pulp and settle to the bottom of the vessel, it is desirable that the consistency should be such that the lighter particles may be kept in suspension by a gentle movement, while the heavier particles fall to the bottom. If the pulp be too thick the metal will remain suspended; if it be too thin the sand will settle with it. Too rapid or too slow motion may produce results similar to the above-named, because by too violent motion the quicksilver will not be allowed to come to rest on the bottom, while if the motion be too slow the coarser sand will not be kept in circulation.

Straining the Amalgam.—From the settlers the quicksilver, which is now amalgamated with a large proportion of the silver in the ore, is drawn off and passed through filters.

In some mills this straining is not performed after every charge of ore, as is the case in others, but only at stated times,

say once in twenty-four hours, or once in three or four days. Under such circumstances a considerable quantity of quicksilver is kept in the settler, sometimes 200 or 300 lbs. This excess of quicksilver, holding the amalgam in solution, is in a highly fluid condition, and when discharged from the settler by means of a tube and cistern afterwards described, it is returned to the pans for further amalgamation, its *charged* condition—that is, having silver already in combination—being considered an advantage, as it is thought to be more active than pure metal in the amalgamating process. In some mills, at a stated hour of each day, the quicksilver coming from the settlers is strained and the amalgam extracted; in others, as the quicksilver thickens or becomes sluggish by the accumulation of amalgam, it is diluted by the addition of fresh quicksilver, and the straining of the amalgam is only made once in several days.

From time to time—as at the end of the month or other given period, or when any special lot of ore has been finished, of which it is desired to know the exact yield—the pans and settlers must be stopped and cleaned up thoroughly. For this purpose the mullers must be raised, the shoes and dies removed from their places, and all the iron work of the pans and settlers carefully scraped with a knife to remove and collect the hard amalgam which attaches itself to such surfaces. In many cases one fourth or even a greater proportion of the total product of amalgam is obtained in this way.

Retorting and Melting. — The amalgam, having been strained in the bags and forcibly pressed, in order to expel as much of the fluid quicksilver as possible, is next subjected to the process of sublimation, by which means the quicksilver is separated from the gold and silver. This is effected in cast-iron retorts.

After retorting, the silver is submitted to the process of smelting in wind furnaces, and when cast into ingots is assayed, and is then ready for the market.

Tailings.—The pulp, after passing from the settlers—in

which, as before described, the quicksilver and amalgam are separated from it—is variously treated in different mills. Frequently the whole mass is allowed to pass through agitators, tubs, or vats of various devices, for the purpose of saving some of the quicksilver and amalgam carried off with it from the settler. In some mills various kinds of concentrators are employed for a similar purpose, and to obtain the heavy undecomposed sulphurets in concentrated form; in other cases, where there is water sufficient and the lay of the land favourable, blanket tables are constructed outside the mill, over which the stream of tailings is allowed to run, and a portion of their valuable contents caught in blankets; and at convenient points dams are constructed for the accumulation of tailings, which, after months of exposure to the influences of the weather, may be again worked over with profit.

The term "tailings" is applied to the residue of sand or pulp that leaves the separator or agitator after the principal portion of its valuable contents has been extracted. The term "slimes" generally applies to that portion of the ore which is crushed under the stamps to an impalpably fine condition, and which usually passes out of the mill without being deposited in the tanks, where the coarser sands are collected for pan treatment. The difference in the value, mechanical condition, and methods of treatment of tailings and slimes makes the distinction between them an important one. That part of the *tailings*, which by grinding in the pan has been reduced to a slimy condition, is sometimes called *pan slimes*, and thus distinguished from *battery slimes*.

Results of the Process.—The ordinary working result obtained by treating the ore as above described, in pans and settlers, varies between 65 and 75 per cent. of the assay value, which, by subsequent treatment, as indicated in the foregoing paragraph, is increased sometimes to 85 or 90 per cent., or possibly a little more.

CHAPTER III.

MACHINERY EMPLOYED IN CRUSHING AND AMALGAMATING.

ROCK-BREAKERS—Blake's and Krom's described—Tulloch's Ore-Feeder —Stamps—The Battery—Mortars—Dies—The Stamp Head—The Shoe—The Tappet—The Guides—The Cam—Number of Stamps—Weight and Drop of Stamps—Quantity of Water used in the Battery—Amalgamating Pans: Wheeler's, Greeley's, Hepburn and Peterson's, Wheeler and Randall's, McCone's—The Combination Pan—Fountain's Pan and Horn Pan—Separators—Agitators—Clean-up Pans—Retorts —Melting Furnace—General Arrangement of Mills.

THE machinery of a mill for the treatment of silver ores by the "wet process," described in the last chapter, consists of rock-breakers and stamps for crushing; vats for settling; pans, for grinding and amalgamation; settlers, for the separation of the quicksilver and amalgam from the pulp; agitators, which are supplementary to the settlers, and save escaping quick-silver; various appliances for the concentration of the residue, or "tailings;" the retort for the sublimation and separation of the quicksilver from the precious metals; and the melting furnace. In addition there must be, of course, the motive power and its auxiliary arrangements.

Rock-breakers, Blake's Patent. — The rock-breaker generally in use is Blake's, Dodge's, Krom's, or some other pattern. In Fig. 3 we have a side view or elevation of the parts in Blake's machine in place as they are presented to view through the side of the frame.

The circle D is a section of the fly-wheel pulley, which should make 225 to 250 revolutions per minute, and will run either

way. E is a pitman, or connecting-rod, which connects the eccentric with the toggles G G, which have their bearings, forming an elbow or toggle joint. H is the fixed jaw; this is bedded in zinc against the frame $\frac{1}{4}$ in. thick.

P P are the chilled plates against which the stones are crushed, and are held back to their place by cheeks I, that fit in recesses in the frame on each side. When worn can be changed to opposite side. J is the movable jaw; this is supported round the bar of iron, K, which passes freely through it, and forms the pivot upon which it vibrates.

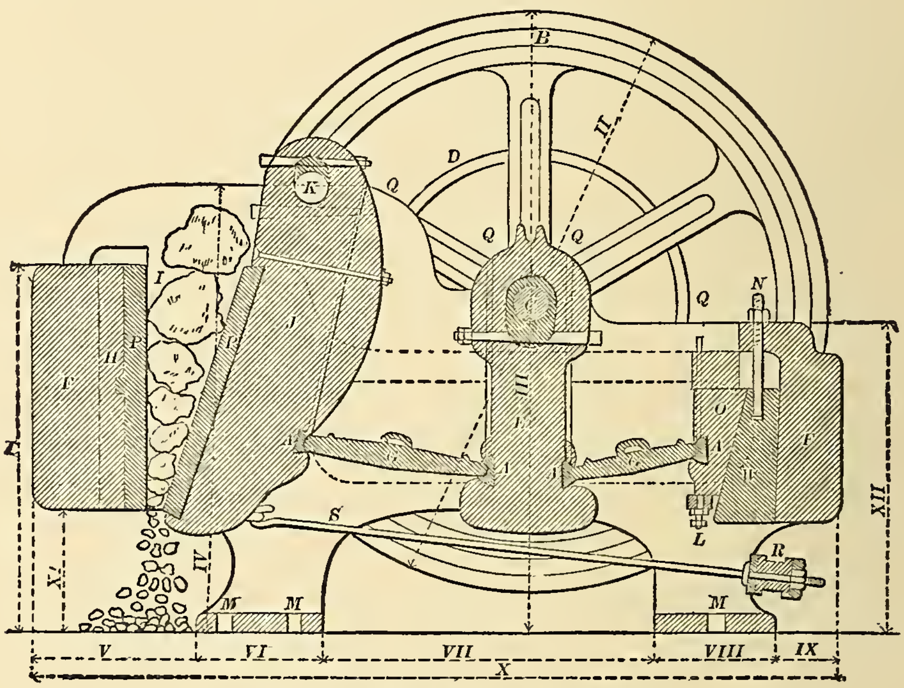


FIG. 3.—BLAKE'S ROCK-BREAKER.

R is a spring of india-rubber, which is compressed by the forward movement of the jaw, and aids its return; M M are bolt-holes; B is the fly-wheel, and D the driving pulley; Q Q Q Q, oiling tubes; A A A A, steel bearings; L, set screw for tightening toggle-block.

Fig. 4 is a plan of the machine. The frame, F F, which receives and supports all the other parts, is cast in one piece, with feet to stand upon the floor or upon timbers. These feet are provided with holes, M M M M, for bolts, by which it may be fastened down if desired. B B are fly-wheels. D is the driving pulley.

Directions for setting up the Blake Stone-Breaker.—1. Place the frame, F F, level on the floor, or on timbers lengthwise.

2. Put in swing jaw, J, tighten the caps on the swing jaw shaft tight enough to keep the shaft from moving, then put on the lock nuts.

3. Put in pitman, E, with large end of key nearest toggle block O. Let it drop on to a block of wood high enough to clear bearings about 6 in., then slide in the shaft C. Notice

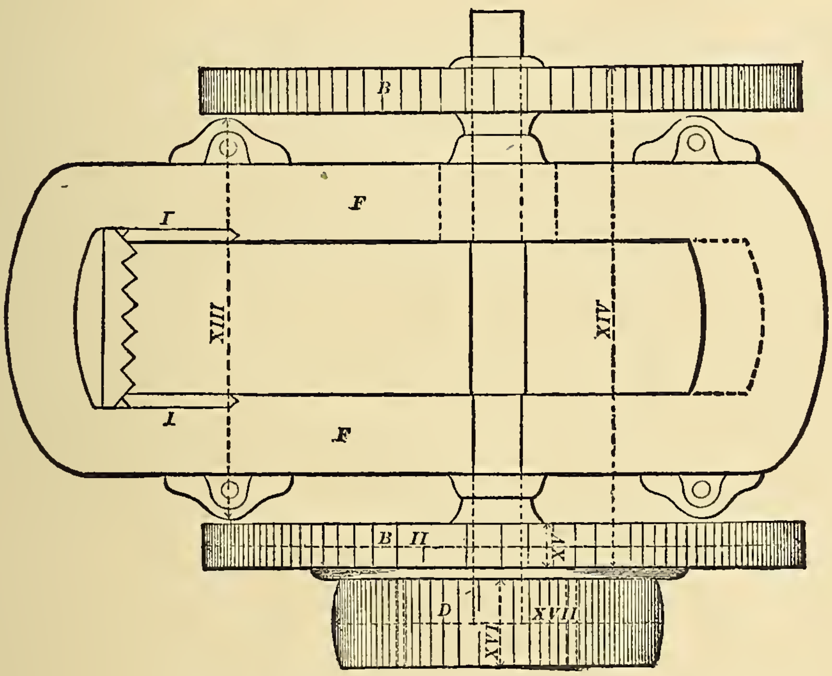


FIG. 4.—BLAKE'S ROCK-BREAKER.

the marks on one end of shaft, so as to get the pulley end on the driving side.

4. Put in the lower box *with thin wood packing*. This packing keeps the key from tightening the lower box to the shaft. Next put in key from the back and tighten set screw.

5. Lower shaft into the bearings and put on caps, having a piece of thin wood or leather under the caps, to keep them from being screwed down too tight on the shaft.

6. Put on fly-wheels according to marks on the shaft. Key them tight to clear the side of bearings about $\frac{1}{16}$ in., and screw on the driving pulley to its place.

7. Put in the two toggles, G G, the longest in front, or

between the swing jaw, J, and pitman, E. Let the wedge, W, be screwed down to the lowest point. By raising or lowering the wedge, W, with nut, N, the size of stone broken is changed; if this will not give the required size change either front or back toggle, keeping the pitman, E, about upright, as shown in the cut. Put in the rod and rubber spring, R, screwing the rubber only tight enough to bring back the swing jaw, J.

8. Tighten the toggle-block, O, with the set screw. Oil bearings by the tubes set in for the purpose. Apply power and the

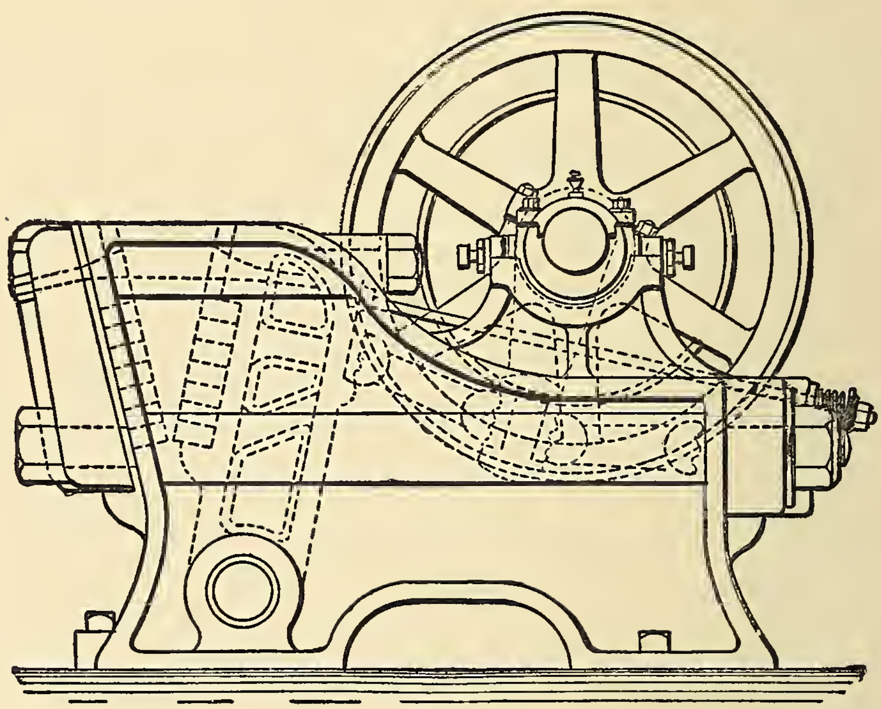


FIG. 5.—KROM'S IMPROVED SECTIONAL ROCK-BREAKER.

breaker is ready for use. Keep iron plugs in the oil tubes to exclude the dust.

9. If the fixed jaw, H, should require to be cast up, use zinc about $\frac{1}{4}$ in. thick. When the jaw plates are worn at the lower ends they can be reversed. If the steel toggle bearings should wear out, they can also be reversed. It will be necessary to drill oil holes on the other side for the oil tubes.

The Krom Rock-Breaker.—S. R. Krom, of New York, has improved the Blake rock-breaker by shortening in his machine the upper tie-bolt, so as to give a better and more

convenient form to the side frame; and the lower tie-bolt is so placed that it receives all the strain due to crushing the ore. In the toggle-abutment, through which the main tie-bolt passes, are recesses around the bolt-holes. These recesses are covered with wrought-iron washers of sufficient strength not to bend under the ordinary strain in crushing the ore, but they will yield to excessive strain. These washers take the place and serve the purpose of the breaking cups in the first machine, but do not fly to pieces, and therefore both ends of the toggle-block

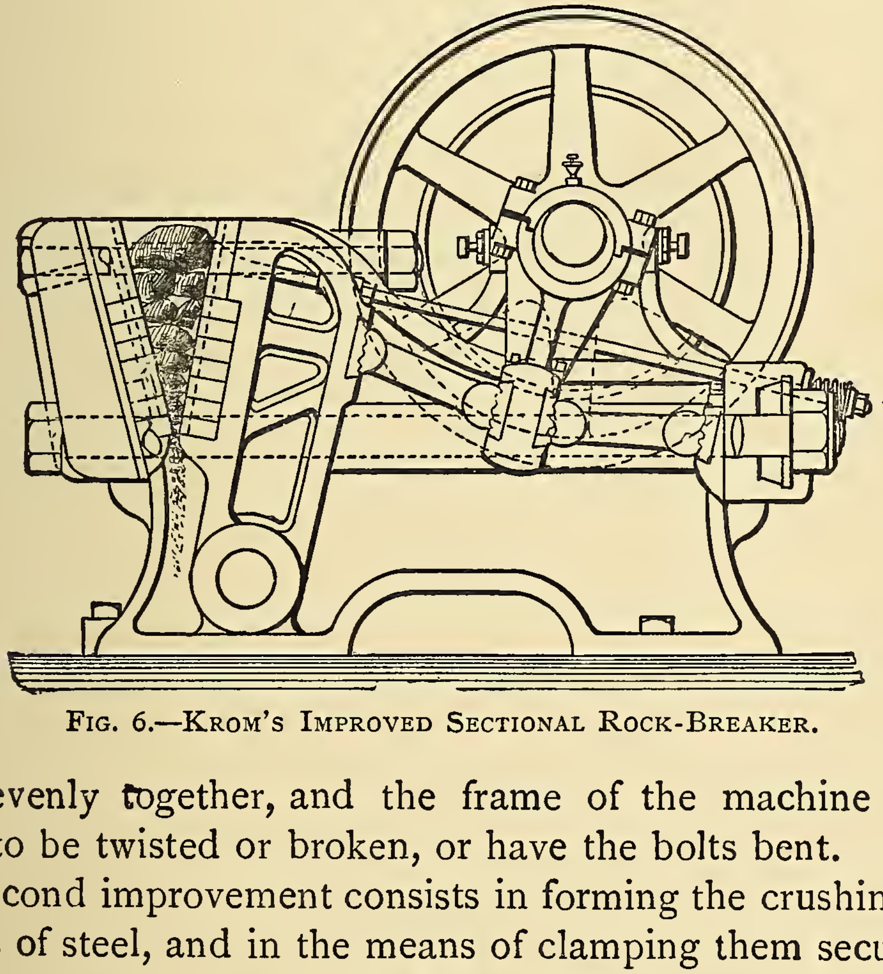


FIG. 6.—KROM'S IMPROVED SECTIONAL ROCK-BREAKER.

yield evenly together, and the frame of the machine is not liable to be twisted or broken, or have the bolts bent.

A second improvement consists in forming the crushing faces of bars of steel, and in the means of clamping them securely in place. These bars are of good steel, rolled exactly to standard sizes, and cut to proper lengths; and the lower bars are hardened to increase their durability. Provision is made for bringing the jaws closer together to compensate for wear. Thin strips of metal are also provided to put behind the bars to keep the wearing-faces in line. The bars, or crushing-faces, can easily be got at by taking the nuts from the bolts, and sliding the stationary jaw forward. (See Figs. 5 and 6.)

A further improvement is made by hanging the jaws on an axis below the crushing-faces, instead of at the top as in the Blake crusher. This manner of hanging the movable jaw gives a more uniform product, and the principle is correct. Since the strain on the jaw is greater at the bottom than at the top, and the motion should be the least where the strain is the greatest.

The last of the improvements consists in employing toggles with rolling ends which work without friction or oil. The bearings are self-adjusting, large, and long, and the machine is constructed for high speed, hard work, and large crushing capacity.

If the feeding of the battery is done automatically, generally some self-feeder is employed, and I describe here one much employed in the mining regions.

Tulloch's Ore-Feeder.—Fig. 7 represents the so-called Tulloch ore-feeder. A is the hopper into which the crushed ore from the stone-breaker is discharged. B, the shaking tray for feeding the ore into the mortar box. C C are suspension links for carrying the tray. D G is a rocking shaft which imparts a rocking motion to the feeding tray through the area E. The lever, I P, is centred at L, and is connected to the shaft, D G, by means of the link, H. By means of the rod, J, the top end of which comes in contact with the tappet of the centre stamp (a hole being bored through the lower guide block of stamps for this purpose), a motion corresponding to that of the stamps is given to the lever I P. O is the framework of the machine, and M is a strong steel spring for giving a sharp recoil to the tray, B, thereby ensuring a proper movement of the ore over the same.

These feeders are exceedingly simple and compact, and save all hand labour in feeding the battery. They can be regulated to feed any quality or quantity of ore. If the dies are deeply covered with ore the stamp does not fall through its full height, and consequently little motion is imparted to the shaking tray. If on the other hand there should be no covering of ore on the

dies the tappet will travel through the full height of its fall, and the rod, J, will move in proportion.

Proper feeding lies between these two extremes, and therefore the rod, J, leading to the lever, I P, should be so adjusted that there shall always be a thin layer of ore on the dies, and thus avoid any loss of power from unnecessary pounding of the ore in the mortar box.

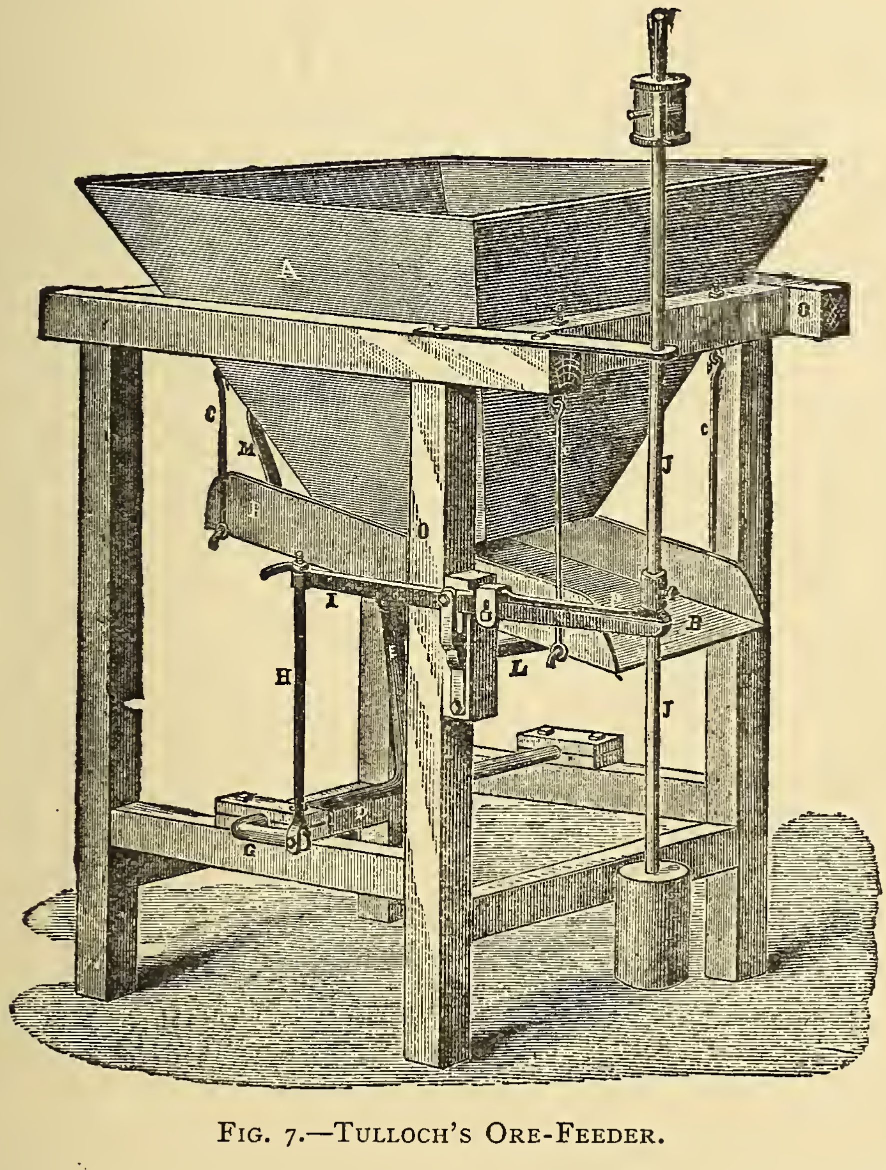
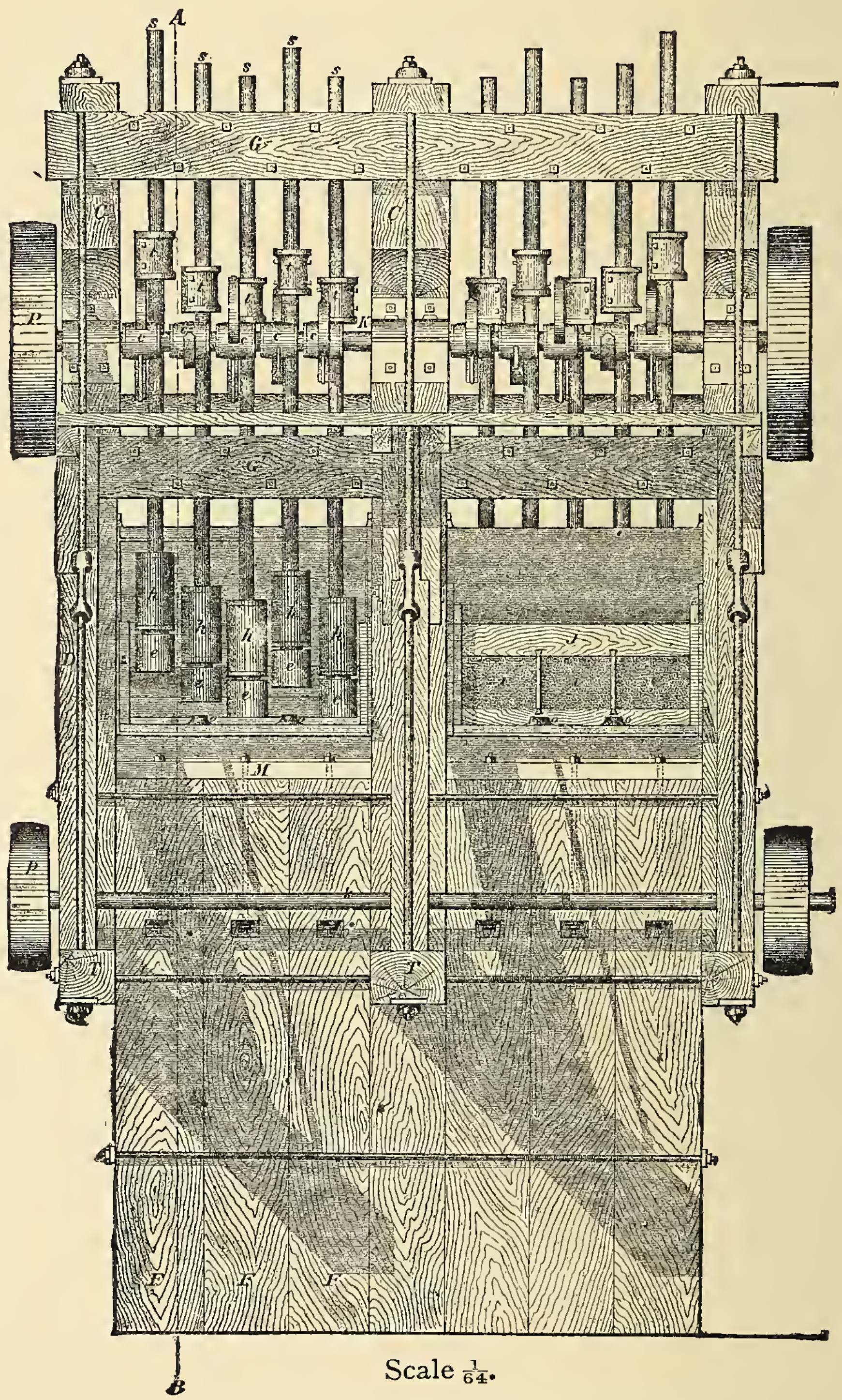


FIG. 7.—TULLOCH'S ORE-FEEDER.

Stamps.—The stamps consist of a series of heavy pestles of iron, which are lifted to a height varying from 7 to 15 in. and allowed to fall upon the ore that is to be crushed. They work in a mortar, or trough, also of iron, into which a constant supply of ore is introduced, and from which the crushed material escapes through openings furnished with closely-fitting screens

as soon as it is reduced to the designed degree of fineness. The mortar is usually rectangular in form, and contains five

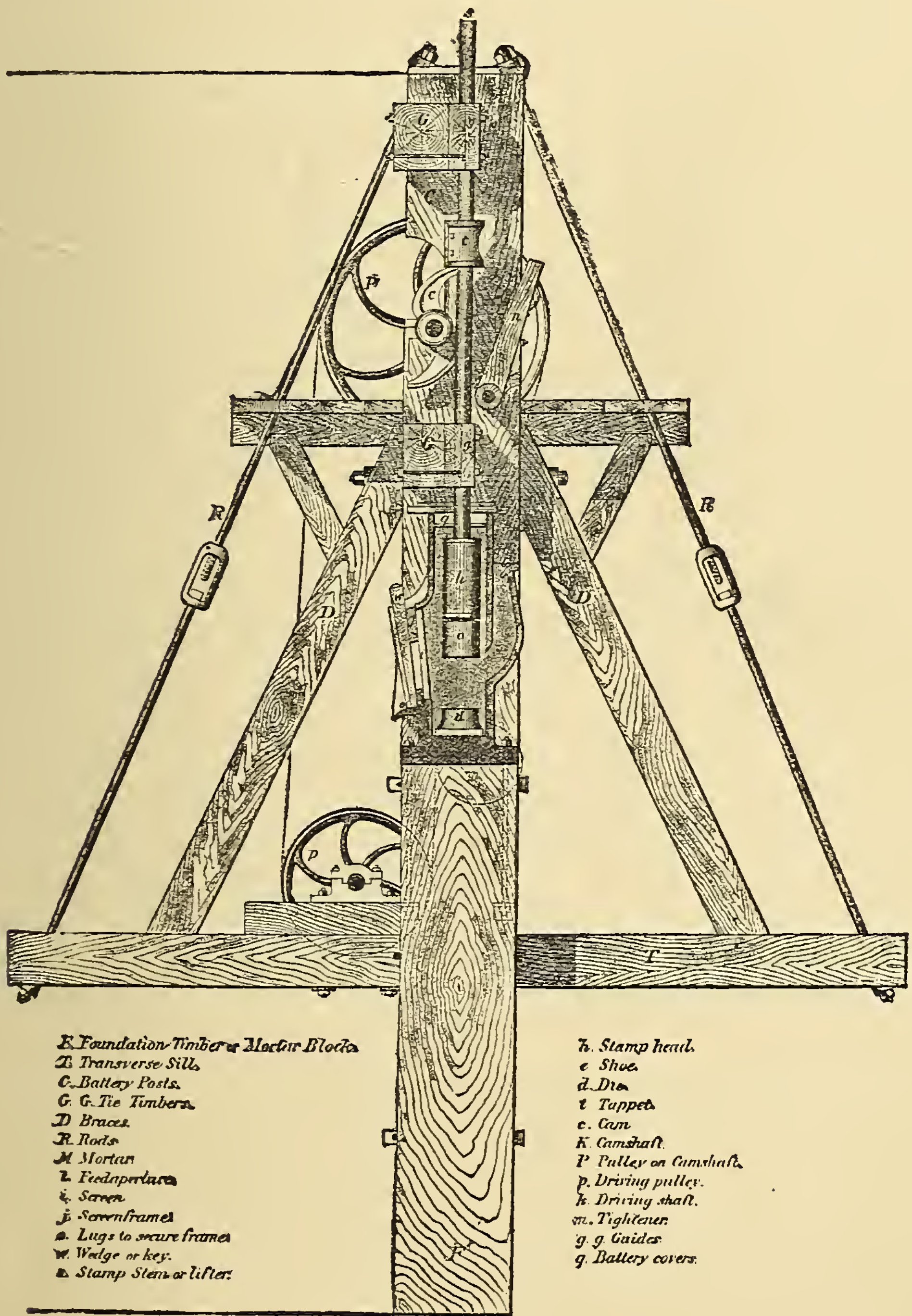


Scale $\frac{1}{64}$.

FIG. 8.—FRONT ELEVATION OF 10-STAMP BATTERY.

stamps, forming a "battery." The mortars rest on a solid foundation, and are established in a substantial framework of timber. The stamps are lifted by means of revolving cams, or

arms of iron, keyed to a cam shaft, which is placed directly in front of the batteries, and which receives its motion from the driving power of the mill. The stamps move vertically between guides that form a part of the battery frame.



Scale $\frac{1}{64}$.

FIG. 9.—TRANSVERSE SECTION OF BATTERY.

The Battery.—Fig. 8 shows a front elevation—and Fig. 9 a transverse section, on the line A B of Fig. 8—of two five-

stamp batteries, the several parts of which are indicated by the table of reference accompanying the drawing.

The foundation for the batteries in stamp mills, which is generally preferred in Nevada, as well as in California, consists of heavy timbers, standing vertically, placed close together, and firmly connected by means of cross timbers and bolts of iron.

The timbers are from 6 to 12 ft. long, according to the character of the ground and the desired height of discharge for the mortar. Sometimes they stand on a horizontal beam, so laid as to serve as the base of two or more batteries, and resting upon the ground, the surface of which has previously been removed and levelled down sufficiently for the whole number of batteries to be placed on a firm bottom. When the foundation timbers are in place, the space about them is packed and stamped as firmly as possible with clay or earth. Where the ground on which the batteries are built is hard, compact gravel, or a firm clayey material, the surface is sometimes levelled off so as to admit of laying the transverse sill timbers, T, of the battery frame, and a narrow pit is then excavated, only long and wide enough to receive the ends of the mortar blocks, and several feet deep, into which the posts or blocks are introduced in a vertical position, their bottom ends resting directly on the ground, without any intervening horizontal timber. The remaining space in the pit may then be compactly filled with clay, which should be pounded or stamped firmly into place. The sill timbers, T, and the battery posts, C, are securely bolted to the foundation timbers. The posts, C, are braced by the timbers, D, and the rods, R, and are connected by the tie timbers, G G″, which also support the guides, *g g′*.

Mortars.—The mortars are now usually placed directly upon the vertical mortar blocks without any horizontal piece intervening, and are secured in their place by bolts shown in the figure. They are made entirely of iron.

The mortar in general use for wet crushing is an iron box, or trough, about 4 or 5 ft. in length and depth, and 12 in., inside, in width, and so cast that bottom, sides, and ends are in one piece. A front and end view of one of the most approved

forms is shown in place, in the drawing of a battery of stamps in Figs. 8 and 9. The feed opening, *l*, is an aperture about 3 or 4 in. wide and nearly as long as the mortar, by means of which the rock is supplied to the stamps. On the opposite side is the discharge opening furnished with a screen, *i*, through which the crushed material must pass. This opening is as long as the mortar, or nearly so, and 12 to 18 in. deep, the lower edge being 2 or 3 in. above the top of the die. In some mortars, especially for dry crushing, the discharge is on both sides, in which case the feed opening is above the screen; but the single discharge is in general use in the Washoe district.

Fig. 10 shows a mortar for wet crushing with double discharge, and fitted with aprons, front and back, having launders for leading off the pulp. The object of the double discharge is to increase the flow of pulp. Screen frames of hard wood are shown held in place by wrought-iron keys, which appear projecting slightly above. In this, as in the single mortar, ore is introduced in the back opening.

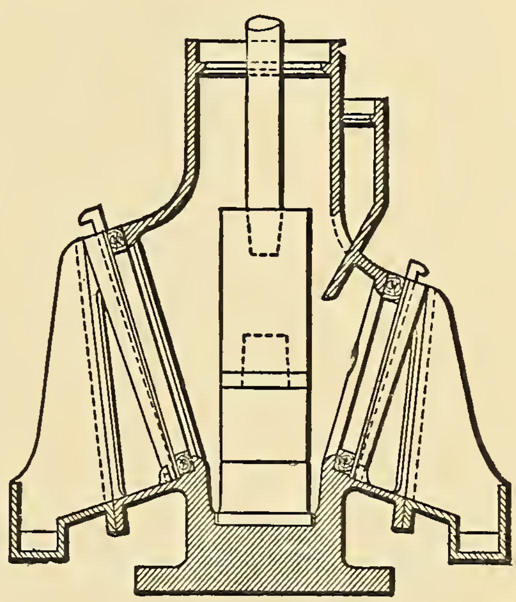


Fig. 10.—Wet Crushing Mortar with Double Discharge.

The Screen is attached to a screen frame, *j*, which is secured in grooves cast in each end of the mortar, and by two lugs, *o*, cast in front of the discharge opening, being held firmly in place by a wedge driven behind it in the grooves just referred to.

Screens are sometimes placed vertically, sometimes inclined, as shown in the figure. The discharge is generally thought to be better in the latter case. Screens are made of fine brass wire cloth, having from 40 to 60 meshes to the lineal inch, of Russia sheet iron, perforated by finely punched holes, varying from $\frac{1}{40}$ to $\frac{1}{24}$ of an inch in diameter. They are attached to the screen frame by nails or screws. The punched plate is pre-

ferred for wet crushing. The wire cloth, though affording more discharging surface, wears out faster, and not only is more liable to break, and so permit large particles to pass through, but frequently stretches, giving meshes of irregular size. A piece of canvas is usually hung before the screen for the crushed ore to splash against as it issues from the mortar, falling thence into the trough below.

Dies.—The mortar is furnished with dies, which are so fixed in the bottom as to receive the blow of the stamp and sustain the wear which would in their absence fall upon the mortar itself. The die is a cylindrical piece of cast iron, corresponding in form to the shoe of the stamp that falls upon it. It is from 4 to 6 in. high. In the bottom of some mortars there are circular recesses fitted for the reception of the dies. In others, to prevent the rock from working in under the die and displacing it, the circular recess is cast with a flange, and the die with a small projection or lug. A groove is also made in the bottom of the mortar, so that the die may be introduced with its lugs dropping into the groove. The die being then turned about 90 degrees, the lugs come under the flanges of the recess and the die is consequently held in place. A simpler (and the most common) form is to cast the cylindrical part of the die on a flat square base, as shown in Fig. 11. The bottom of the mortar is also made flat, and the dies dropping in rest on their bases, which just fill up the space in the bottom of the mortar. The corners of the bases of the dies are bevelled off so as to allow the insertion of the point of a pick, by which means they can be taken out when necessary.

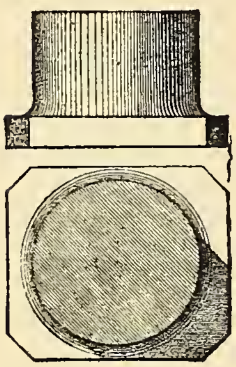

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FIG. 11.—THE DIE.

In addition to the dies, plates of iron $\frac{1}{2}$ in. thick are sometimes applied to the sides and ends of the mortar exposed to constant wear, which, like the dies, can be taken out and renewed when necessary. The top of the mortar is covered by two pieces of plank, cut so as to fit closely, and resting on flanges cast on each end. Semicircular recesses, cut opposite

each other on the adjacent edges of the two pieces of plank, afford a passage for the movement of the stamp stems.

The Stamp.—The stamp consists of a stem or lifter; a head or socket, attached to the lower end of the stem, and furnished with the shoe, a movable part which sustains the force of the blows and the wear of the operation; and the tappet, by means of which the revolving cam lifts the stamp for its fall. The stem is a round bar of wrought iron, about 3 in. in diameter, usually turned in a lathe. Its length is 10 or 12 ft. Its lower end is slightly tapered, and corresponds in form to a socket or conical hole in the upper part of the stamp head. The rest of the stem is usually made round throughout its entire length, the method, now in general use, of attaching the tappets to the stems not requiring any modification in the form of the latter, as was formerly the case.

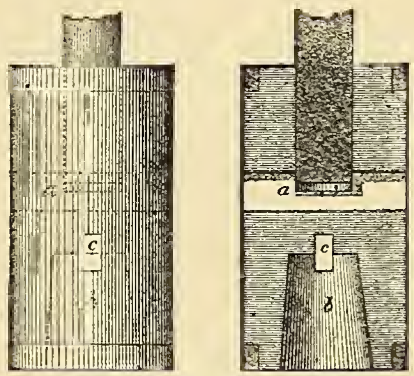

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FIG. 12.—THE STAMP HEAD OR BOSS.

The stamp-head, illustrated by Fig. 12, is a cylindrical piece of tough cast iron about 8 in. in diameter and 15 in. high. In its upper end is a socket, shown by dotted lines, corresponding with the axis of the cylinder and conical in form, designed to receive the slightly tapering end of the stem, to the dimensions of which it must be adapted. This conical hole or socket is about 7 in. deep. At its bottom is a hole, or key way, *a*, passing through the head at right angles to the cylindrical axis, by which passage a key may be driven in to force the head from the stem when necessary.

To attach the stamp-head to the stem the latter is placed in its position between its guides, and the head standing immediately under it. The stem being dropped enters the socket, and a few blows of the hammer drive it in with sufficient force to cause the head to be raised when the stem is lifted. The stem and head, being suffered to drop together a few times, become firmly connected. In the lower end of the head is a

similar head or socket, *b*, but larger than the upper one, likewise tapering or conical in form, made to receive the stem or shank of the shoe, which is thus connected with the head in similar manner ; a rectangular hole or passage, *c*, through the head at the end of this lower socket, permits the removal of the shoe in the same way as the stamp-stem is forced out from the upper socket. A stout wrought-iron hoop encircles each end of the stamp-head, being fitted and driven on when hot and allowed to shrink in place. This practice is now mostly done away, and a superior quality of iron employed for these portions of the stamp.

The Shoe in common use in these mills is a cylindrical piece of cast iron about 8 in. in diameter and 6 in. high, above which is a shank or stem, the base of which is 4 in. or 5 in. in diameter, tapering in form and about 5 in. high. It is made of the hardest white iron. It is attached to the head in a manner somewhat similar to that just described for connecting the head and the stem, but is wedged on by means of strips of pine wood, These strips, which are cut about as long as the stem of the shoe, $\frac{1}{4}$ in. thick and about $\frac{1}{2}$ in. wide, are placed around the stem of the shoe and tied with a piece of twine, as shown in Fig. 13. They must be thick enough to wedge the stem of the shoe firmly in its socket, without allowing the head to come in contact with the body of the shoe. When the shoe is ready to be fixed to the head it is placed in proper position with the stem of the shoe directly under the socket of the head, and the stamp and head are then allowed to drop upon it. If necessary, a few blows of the hammer must be struck upon the top of the stamp stem. The whole may then be raised, the shoe keeping its place, and suffered to fall repeatedly until the shoe is firmly established in the socket. During this operation a piece of plank is interposed between the die on the bottom of the mortar and the shoe for the latter to strike upon. Whenever a shoe has been worn out it may be removed from the socket by driving the key into the key-way, *c*, and forcing it off.

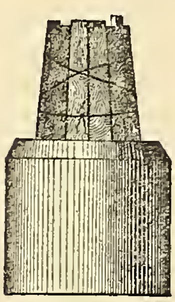

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Fig. 13.
The Shoe.

Care is required that the shoe does not become so thin as to permit the head to sustain undue wear, and so become weakened. Shoes should be removed when worn down to 1 in. of thickness.

The Collar, or Tappet, is a projecting piece, firmly secured to the upper part of the stem, by means of which the revolving cam may lift the stamp and let it fall upon the substance to be crushed. Tappets vary in form and method of attachment to the stem, but that which seems to combine the greatest number of advantages, and to have been most generally adopted in California and Nevada, is the one known as Wheeler's gib-tappet. Fig. 14 shows an elevation and vertical section of this contrivance. It is a piece of cast iron, cylindrical in form, about 8 in. in height and diameter, hollow at the centre, so as to receive the stamp stem. To secure the tappet to the stem there is a gib, *g*, about 2 in. wide and nearly as long as the tappet, having its inside face curved so as to correspond in form to the circular hole through which the stem passes. The gib being fixed in its place in the tappet, and the latter being put upon the stem, it is pressed against the stem by means of two keys, *k k*, driven into the key-ways, with force sufficient to hold the tappet and stem firmly together, and prevent slipping between them. This is found to be a very effective method of securing the tappet, while permitting it to be fixed at any desired point on the stem, according to the wear of the shoe. The stem is uniform in size, and the work of cutting facings, screw threads, and key seats on the stem, required by other methods in use elsewhere, is thus avoided.

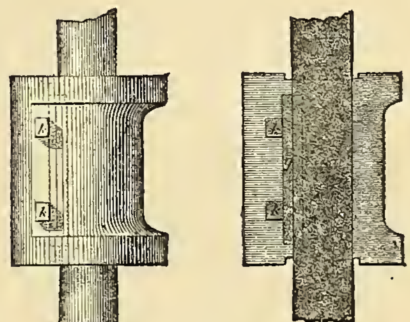

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FIG. 14.—THE TAPPET.

The rotary motion of the stamp imparted by the friction of the cam against the tappet is in very general use in Nevada. This is one of the advantages offered by the use of round shoes, stems, and tappets. The revolving cam, meeting the tappet and raising the stamp, causes it while being lifted to make a

partial revolution about its vertical axis, which rotary motion being continued during the free fall of the stamp produces a grinding effect between the shoe and die upon the substance to be crushed. Not only is the effective duty of the stamp at each blow increased in this way, but the shoe wears down much more evenly than when it falls without such rotary motion.

Guides.—The stamp is held vertically in its movement by guides, between which the stem passes. These were formerly made of iron, but wood has been almost universally substituted for iron in Nevada and California. One set of guides is placed below the tappet, about a foot above the top of the

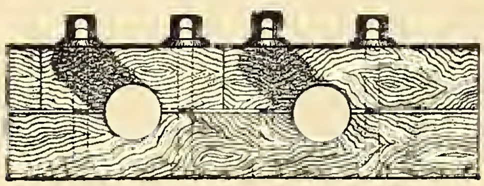

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Fig. 15.—The Guides. Plan.

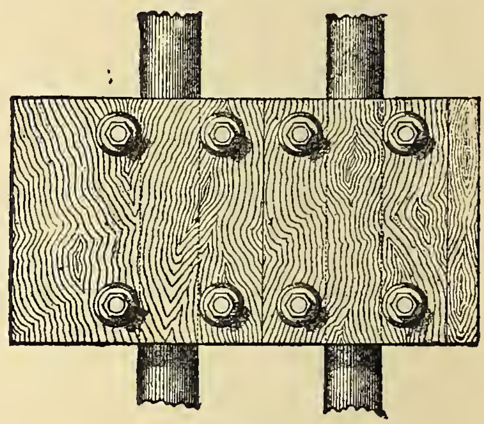

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Fig. 15*a*.—The Guides. Elevation.

mortar; the other set is placed near the top of the stem, so that 6 in. or a foot of the latter may project above the guides. They are supported by the cross timbers, or ties, G G″, which form a part of the battery frame connecting the two uprights or posts. They are usually made of pine, though hard wood is preferred, and are from 10 to 16 in. wide. One part of the guide is made in a single piece for the whole battery, bolted to the cross timber; the other part may be in one piece, like the first, or as in Fig. 15, cut into as many pieces as there are stamps in the battery, which are then secured to the corresponding part by bolts. In each part are cut semicircular recesses, and when the two parts are put together so that the recesses correspond, the holes or stemways for the reception of the stamp stems are formed. When the guides are so

worn by friction as to permit too much motion of the stems, they may be dressed down on their adjacent faces, by which means the recesses are reduced to nearly the proper dimensions.

Cams.—The cam is a curved arm fixed to a shaft, which is so placed in front of the battery that, by the revolution of the shaft, the cam is brought into contact with the tappet of the stamp stem, causing the latter to rise to a height determined by the length of the cam, and to fall at the moment of its release from such contact.

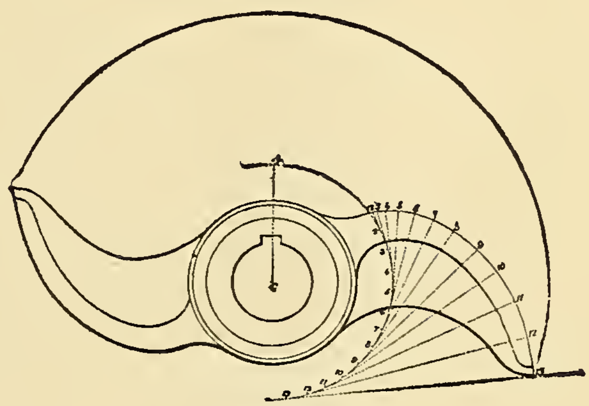

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Fig. 16.—The Cam.

In Nevada the cams are made of tough cast iron, and are usually double-armed, that is, having two arms attached to one central hub. Figs. 16 and 17 show the form of cams most generally in use. In Fig. 17, *a* is the hub, *b b* are the arms, *c* is the face, and *d* a strengthening rib.

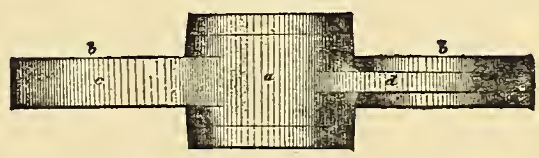

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Fig. 17.—The Cam Face.

The proper curve of the face of the cam, in order that it may perform the desired duty with the least friction, is the involute of a circle, the radius of which is equal to the distance between the centre of the cam shaft and the centre of the stamp stem. This produces a line for the face of the cam which meets the various requirements better than any other. The bottom of the tappet is constantly perpendicular to the radius of the curve of the cam. The tappet, and with it the stamp, is lifted vertically and uniformly, so that the lift of the stamp is always proportioned to the revolution of the cam shaft.

The cam curve may be constructed on paper by means of tangents, as shown in Fig. 16. If *c* represents the centre of the cam shaft, and *c r* the distance from the centre of the cam shaft to the centre of the stamp stem, the circle described about *c*,

with *cr* as a radius, is the developing circle of the involute. The distance, representing the height to which the stamp is to be lifted, is laid off upon the circumference of this circle, as from the point 1; which distance is subdivided into a convenient number of equal parts, determining, as in Fig. 16, the points 2, 3, 4 . . . . . 13. From each one of these points in the circle a tangent is drawn, on which is laid off a distance equal to the length of arc between the point 1 and the point from which the tangent is drawn. All the points thus determined in the tangent lines are points in the cam curve, and may be connected, as shown in the figure, thus producing the line for the face of the cam.

In practice the line of curvature is produced by cutting from a thin board a circular piece, the radius of which is equal to the horizontal distance from the centre of the cam shaft to the centre of the stamp stem. At a given point on the periphery of the circular piece is fixed one end of a thread, which must have the length of the greatest desired lift of the stamp, and to the other end of which is attached a pencil point.

The circular piece, with the attached thread wound on the periphery of the circle, is laid on a smooth board on which the line is to be traced, and the thread, being constantly stretched to its furthest reach, is unwound until it forms a tangent to the circle at the point where the other end is attached. The line described by the pencil point is the desired curve.

Some builders slightly modify this curve, giving to the cam arm a greater curvature near each of its ends, in order that the cam, in its revolution, may come in contact with the tappet at the least practicable distance from the cam shaft, where the concussion is less than at a greater distance, and to diminish the friction between the extreme end of the cam and the face of the tappet. The face of the cam is 2 or 2½ in. wide. Its extreme end is fashioned so as to correspond to the outer edge of the tappet, which is circular. The cam is placed as near the stamp stem as practicable, without coming in contact with it. The cams are caused to revolve by means of the cam shaft, to

which they are secured by one or sometimes two keys or wedges.

The Cam Shaft is a round shaft of iron, which is smoothly turned and finished, having one or two key-seats or grooves cut in it lengthwise, for the purpose of securing the cams in their places. The shaft rests in boxes, which are usually supported by shoulders cut on the upright posts of the battery frame. Cam shafts vary in diameter from 4 to 6 or 7 in., according to the number of cams to be fixed upon them and the weight of the stamps to be raised. In some mills a single cam shaft is made long enough to carry all the cams for as many batteries as there may be. In Nevada and California, however, short cam shafts are in general use, a separate shaft being employed for each battery, or in most cases one shaft for two batteries. Separate cam shafts are preferred on account of the independence of each battery, so that if one be stopped by any accident to the cams or the stamps, or for repairs of any kind, the operation of the others is uninterrupted. Each shaft, in such case, is driven by its proper pulley, which receives its motion, by means of belting, from a countershaft. The cam shaft is set in motion by applying the tightening pulley to the belt.

The Number of Stamps in each battery is commonly four or five, the latter number being generally preferred. The order in which they are allowed to drop is not always arranged in the same manner in different mills; but the desired conditions are that the weight of the stamps to be raised may be uniformly distributed on the cam shaft, so that the weight of metal lifted may be as nearly as possible the same at any moment of the revolution, and that each stamp may fall effectively upon the material to be crushed, and by the force of its blow aid in the proper distribution of the stuff among its neighbouring stamps. If the stamps are allowed to rise and fall in regular succession from one end of the battery to the other, the material is usually found to accumulate at one end, and the effective duty of all the stamps is greatly diminished.

The order must therefore be varied. In a five-stamp battery a common arrangement is to let fall first the middle stamp, then the end stamp on the right, then the second stamp on the left, then the second stamp on the right, and finally the end stamp on the left. The order in which the stamps are to fall being determined, it is carried into effect by fixing the cams on the shaft in such position that each cam, by the revolution of the shaft, will lift its respective stamp at the desired moment. For this purpose the key-seats cut in the hub of the cam must be determined with care; one common key-seat being cut on the cam shaft, when the desired position of any given cam has been ascertained, the key-seat in the hub is cut to correspond with that of the shaft.*

The Hangers, or Props.—When it becomes necessary to hang up a stamp so that the cam may revolve without reaching the tappet, it is supported by a prop or stud, N, which is shown in Fig. 9. The lower end of the studs, of which there is one for each stamp, is pivoted on a small shaft fixed across the battery from end to end, resting in boxes which are secured to the uprights. Each stud is just long enough to support the stamp when placed under the tappet, at a height which is about an inch above the highest lift given by the cam. To bring the end of the stud into this position when desired, the workman lays a smooth stick on the face of the cam as it is rising to the tappet, and holds it there while the stamp is lifted. The stick is as wide as the face of the cam, long enough to be held conveniently, and an inch and a half thick at the end which comes between the cam and tappet. By this means the stamp is raised high enough for the stud to be put in place, which being done the stamp is supported above the reach of the cam. To set it again in motion the operation is repeated, the stud being withdrawn at the moment when the stick on the face of the cam has lifted the stamp clear of its support.

Weight and Drop of Stamps.—In Nevada the stamps in most general use weigh between 700 and 850 lbs. They are

* See my "Metallurgy of Gold," p. 35.

usually run at about 70 or 80, sometimes 90, or even 100 blows per minute; they drop from 7 to 10 inches, according to speed, the greater number of blows per minute requiring shorter lift. In reducing the quartz of the Comstock lode by wet crushing, discharging through a No. 5 or 6 screen, the average duty is about 2 tons in 24 hours. In some mills it is said to reach 3 tons per day. Much of the effectiveness of the stamps depends on the degree of care devoted to keeping the working parts in good condition, and on the regularity with which they are supplied with ore. This is sometimes done by hand labour, the rock being shovelled in as fast as it is crushed and discharged. In some mills, however, automatic feeders are employed, which give satisfaction.

Quantity of Water used.—The quantity of water consumed in the batteries varies with the character of the ore and the degree of fineness to which it is crushed. Usually, in the mills of the Washoe district, the consumption is between 250 and 300 cubic ft. per ton of rock treated, or from one-third to one-half of a cubic ft. of water per stamp per minute; but this includes the water used in the pans which does not pass through the batteries.

At the Petaluma mill the supply tank contains 4,400 cubic ft. of water, which is sufficient for eight hours' work of full duty. The mill has 24 stamps, which crush 55 tons of rock per day, discharging at only one side of the mortar through a No. 6 punched screen. The consumption in this instance is equal to 240 cubic ft. of water per ton of rock, or $\frac{38}{100}$ of a cubic ft. of water per stamp per minute. Making a due allowance for a portion of the water used in amalgamation, without having passed through the batteries, the quantity actually used in crushing in this mill does not exceed one-fourth, or possibly three-tenths of 1 cubic ft. per stamp per minute.

The method of measurement in the delivery of water is by "miners' inches." A "miner's inch" is the quantity of water that will pass through an orifice 1 in. square in the side of the measuring box, under a head usually of 6 in. The measure-

ment is not uniform throughout the country, as different heads are used in different places. Generally, however, in California the aperture is made 2 in. deep and as long as need be in order to furnish the requisite number of inches, and the water in the measuring box, which is at one side of the supplying flume, is allowed to attain a height of 6 in. above the centre of the orifice.

The quantity of water that will pass through an orifice of 1 in. square in the side of the box, under a head of 6 in., determined by multiplying the area of the orifice by the theoretical velocity $\sqrt{2\,gh}$, and taking two-thirds of the product as effective discharge, is ·02633 cubic ft. per second, 1·578 cubic ft. per minute, and 94·68 cubic ft. per hour. The mill just referred to uses 5 in. of water. Assuming that its measurement is uniformly in accordance with the above conditions, the amount delivered in 24 hours is 11,361 ft.; equal to about 206½ cubic ft. per ton of rock treated. Taking the operation of this mill as a criterion, 1 in. of water is a supply for five stamps, including the quantity required for amalgamation as well as for crushing.

The following reckoning is usually taken for the consumption of water in different parts of the mill :—

For boiler . .	7½ gallons per horse-power per hour.
,, each stamp .	72 gallons per hour.
,, ,, pan . .	120 ,, ,,
,, ,, settler .	60 ,, ,,

If the water used in the battery, pans, and settlers be run into settling-tanks, it can be re-used with a loss of about 25 per cent.

Amalgamating Pans.—There are in use various kinds of pans which, although resembling each other in general character, present important differences in details. These differences have been gradually developed since the first introduction of the common pan, each pan being specially designed to meet some one or more of the various requirements of an efficient machine. The object of inventors, in the main, has been to produce grinding surfaces of most effective form.

securing greatest uniformity of wear with economy of power; to obtain the most favourable conditions for amalgamation, depending mainly on the free circulation of the pulp, the uniform and thorough distribution of the quicksilver, and the proper degree of heat; and to combine with these requirements simplicity and cheapness in construction, facility in management and repair, large capacity, and economy of time, labour, and materials in the performance of duty. The attempts made to obtain these results have met with varied success, the different devices of any one pan sometimes securing a high degree of excellence in certain details at the cost of it in others.

Among the differences in characteristic features of pans, the most noticeable is that of the bottom and the grinding surfaces, some being flat, and others variously curved; other details, of more or less importance—such as the construction of the muller and the method of attaching it to the driver, the form of the shoes and dies, the means of fixing them in place, providing for the heating of the pulp and for its circulation during the grinding and amalgamating process—vary considerably in the several patterns.

The opinions of practical mill-men are somewhat divided regarding the comparative advantages of the different forms of pan bottoms. The prevailing opinion, however, is that flat-bottom pans are the best. While other forms of grinding surfaces may possess superior advantages theoretically, their greater efficiency, in practice, is often lost by the unequal wear of the surface of the muller, usually resulting from the difficulty of keeping the other parts of the machine, on which the grinding surfaces depend, in perfect order. The various parts of the flat muller are simpler in form, more easily handled, and more conveniently replaced when worn out. While the flat-bottomed muller involves the expenditure of more power in carrying its load of thick pulp, this disadvantage is counterbalanced, in the opinion of some, by the more complete distribution of the quicksilver and the consequently more perfect amalgamation.

The flat-bottomed pans of Varney and of Wheeler, and that of Hepburn and Peterson with conical bottom, have been

widely used during several years past. Some improvements have been added to them lately, and they are still held in high esteem by mill-men. Other makers have introduced new pans, the characteristic features of which are great capacity and simplicity of construction. Such are the large flat-bottomed pans of McCone, Horn, and Fountain, which in their mechanical details seem to combine some of the best results of the experience that has been gained since pan amalgamation was introduced, and by their enlarged dimensions to have the capacity for treating, in the same or nearly the same period of time, a charge three or four times as great as that treated by any of the pans formerly in use.

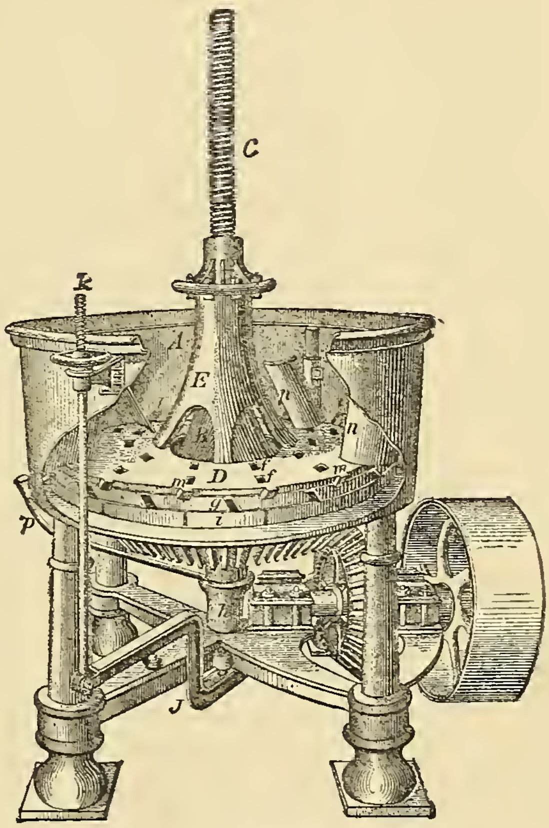
In the following pages a few of the pans that have been used and are still in most favour will be described briefly, but with sufficient detail to indicate their most characteristic differences.

Figs. 18, 20 and 21, present views of three well-known pans. They show the three different forms of pan-bottoms—the flat, conical, and conoidal. The flat-bottomed pan, Fig. 18, known as Wheeler's amalgamator, is perhaps in more general use than either of the others, although Hepburn and Peterson's pan is in great favour among many mill-men.

Wheeler's Pan.—The Wheeler pan of ordinary size is about 4 ft. in diameter at the bottom, and 2 ft. or little more in depth. The general arrangement of the several parts of the machine may be readily seen by a glance at the drawing.

A is the rim of the pan, in the centre of which is the hollow cone, B, rising from the bottom, with which it is cast in one piece. Through this cone the vertical shaft, C, passes, which, being driven by the gearing below the pan, gives motion to the muller, D, by means of the driver, E, which is keyed to the shaft, C. The muller is provided on its under side with shoes, *g*, that form the upper grinding surface. The form of the shoes is shown in Fig. 19. They are attached to the muller by means of two lugs or projections, *f f*, which are received in corresponding apertures in the muller plate, and securely wedged with pieces of wood. The lower grinding surface is

formed by the dies, *i*, which are usually four or eight in number, covering the greater portion of the pan bottom, and secured to it in a manner similar to that by which the shoes are fixed to the muller. There is a radial slot, or space between the dies, which is commonly filled with hard wood. Below the bottom is a steam chamber for heating the pulp. The vertical shaft or spindle, C, rests in a step box, *h*, to which oil is conveyed by the pipe, *p*. A vertical pin passes downward through the centre of the step box, in contact with the shaft, and resting its lower end on the lever, J. This lever may be raised or lowered slightly by the hand wheel on the rod, *k*, thus raising the muller from the dies, if desired. The shaft, C, is also furnished with a screw by means of which the muller may be raised up entirely above the rim of the pan for the purpose of cleaning up or changing the shoes and dies. The hoisting apparatus required in the absence of this screw is thus avoided.



Scale $\frac{1}{48}$.

Fig. 18.—Wheeler Pan.

A, Pan Rim.
B, Central Cone.
C, Central Shaft.
D, Muller.
E, Driver.
f, Aperture for attaching Shoes to Muller.
g, Shoes.
h, Step Box.
i, Dies.
J, Lever for raising Muller
k, Rod for moving Lever.
l, Projection on Pan Rim.
m, Similar Projection or Muller.
n, Wings attached to Pan Rim.
p, Oil Conveyor.

In order to give an upward current or movement to the pulp there are inclined ledges, *l*, on the rim of the pan; and smaller ledges, *m*, on the periphery of the muller, but inclined in the opposite direction. The pan is also provided with wings, or guide plates, *n*, four in number, which serve to direct the moving pulp toward the centre. They are fitted into and may

be removed at pleasure from a T-shaped projection on the pan rim. The muller is caused to make usually about 60 revolutions per minute. It requires from $2\frac{1}{2}$ to 3 horse-power. Its ordinary charge is 800 to 1,000 lbs., but in some mills larger charges are worked. The capacity of the pan is sometimes increased by adding a rim of sheet iron so as to increase the height of the side. The treatment of the charge usually requires four hours.

The shoes and dies usually wear out in from three to six weeks, though they are made to last longer in some mills, their duration depending greatly upon the order in which the pan and all its principal working parts are kept. On this condition the economy in the wear of iron and the efficient operation of this and other pans chiefly depend. Neglect in oiling the working parts of the running gear is apt to cause unequal wear ; the vertical shaft gets loose and out of line, the grinding surfaces cease to work together evenly, and the efficiency of the pan is greatly impaired, while the cost of working is very much increased. Mill-men generally prefer a shoe and die of moderate rather than excessive hardness. The former wear out faster, but are thought to grind more efficiently. They are usually cast of an equal mixture of white and soft iron.

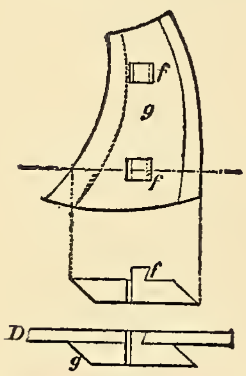


FIG. 19.—SHOE IN WHEELER PAN.

Greeley's Pan.—A pan known as Greeley's, which is used in some mills, and which is highly spoken of, possesses the essential features of Wheeler's, but differs from it in minor details, and has larger capacity. In the Petaluma mill, where ten of these pans are employed, the charge of ore consists of 2,200 lbs.

The bottom of the pan, like Wheeler's, is flat, and has a steam chamber. The dies are cast in four quadrant-shaped pieces. In the middle of each piece, on the upper side, is a radial groove or canal, leading from the centre to the circumference, which permits the free circulation of the material. A

similar space is left between the two adjacent edges of the several pieces. The dies are secured to the bottom of the pan by a dovetailed or wedge-shaped projection, 5 or 6 in. long and from 3 to 4 in. in width on the under side of each piece, which fitting into a similar recess in the pan bottom holds them fast.

The muller is a circular plate, cast separately from the driver, to which, for use in the pan, it is connected by means of four short uprights or legs, that are bolted both to the driver and the muller. The shoes are attached to the muller plate in a manner similar to that by which the dies are secured to the bottom. On the upper side of each shoe is a projection, wedge-shaped in horizontal section, 5 in. or 6 in. long, and from 3 in. to 4 in. wide, which fits into an aperture of corresponding form in the muller plate, and so placed that the smaller end of the projection follows the larger end in the direction of revolution; so that the motion of the muller tends to fix the shoe more and more firmly in its place.

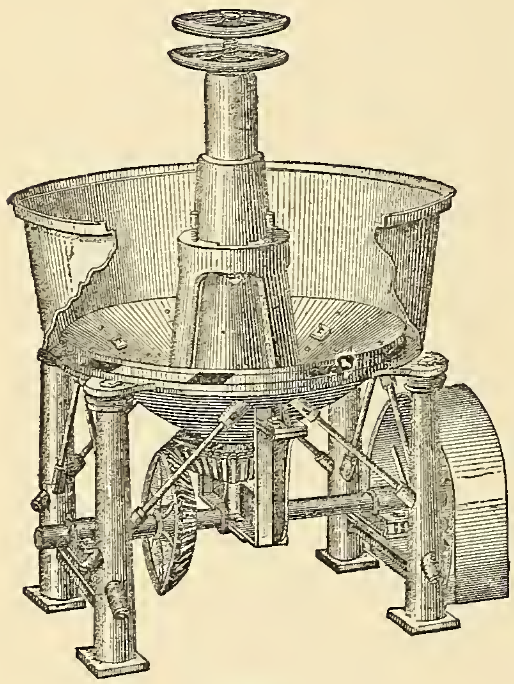
The muller, when in place, is raised and lowered, not by a lever below the step box, as in the case of the Wheeler pan, but by a screw which passes through the hub of the driver, and rests with its lower end on the top of the driving shaft. A hand wheel at the upper end of the screw serves to turn it, raising or lowering the muller, and another hand wheel lower down acts as a jam nut to keep the muller at the desired height. When the muller is in motion it may be raised or lowered by arresting the last-named wheel.

To clean the pan up, the muller plate is lifted entirely out by means of a block and tackle. In some mills this is conveniently supported on a truck which moves on a railway at a suitable height above the pans. By this means the truck can be brought into position above any one of the pans from which it is desired to raise the muller, and the hoisting apparatus thus applied.

Guide plates are used for directing the pulp, the circulation of which, with these contrivances, is very active, the pulp passing from the periphery of the pan at the surface downward towards the centre, producing the surface of a hollow cone, through the aperture at the base of the driver and outward

through the channels, and between the surfaces of the shoes and dies to the circumference, where it rises to repeat the process. The legs or standards of the driver, connecting it with the muller plate, promote this circulation by forcing the pulp to the centre and downward between the shoes and dies. The pan is said to require about four-horse power.

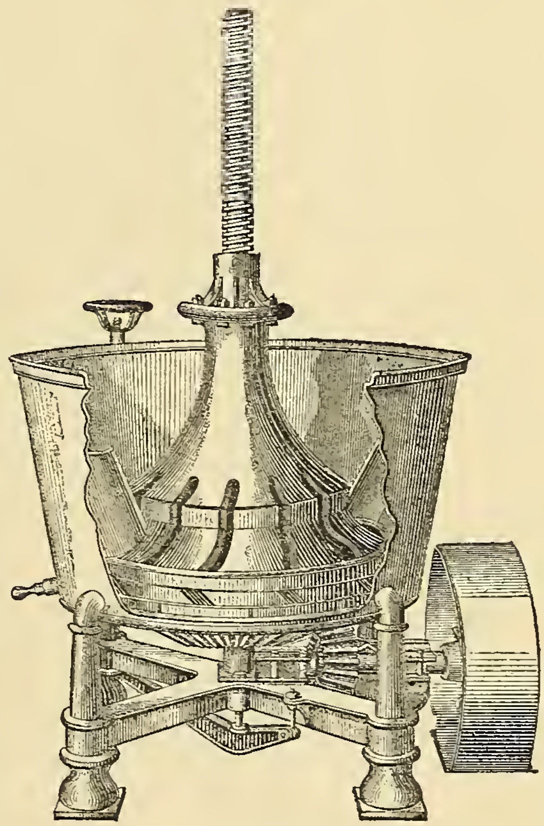
Hepburn and Peterson's Pan (Fig. 20).—The bottom of this pan has the form of an inverted cone, inclining toward the centre, as may be readily seen in the figure. The bottom is covered by four dies of corresponding form, which are secured in a manner similar to that employed in the other pans already described. There is no steam chamber in the bottom, steam being introduced directly. In the centre of the pan a hollow pillar rises, through which the driving shaft passes. The form of the muller corresponds with that of the bottom, and at the centre it has an upright hollow cone, by means of which it is connected with the hub or driver. The under side of the muller is furnished with shoes, between which, when attached to the muller, there is a channel or radial passage left for the circulation of the pulp. The muller also contains radial grooves between the shoes, so that, when the latter wear down, the channel may still be large enough to permit an easy movement of the material. The muller is raised or lowered by means of a screw and movable nut at the top of the hub, the screw resting on the top of the driving shaft, to which the hub is keyed. The circulation of the pulp in this pan is effected without the use of wings or guides, such as are commonly em-



Scale $\frac{1}{48}$.

IG. 20.—Hepburn and Peterson's Pan.

ployed in other pans for this purpose. When the muller is in motion the pulp, passing between the grinding surfaces, from the centre to the circumference of the pan, descends again by its own weight towards the centre on the upper side of the muller; a movement promoted by the conical shape of the muller plate. In the use of guide plates or wings to aid the circulation there is sometimes a difficulty experienced in the tendency of coarse sand to settle and pack firmly, if the pan is stopped for a little while, and giving much trouble in starting again. By thus dispensing with the use of wings some inconvenience is avoided. The charge of the pan is about 1,500 lbs., usually working four hours on a charge. It runs at 60 or 70 revolutions per minute.



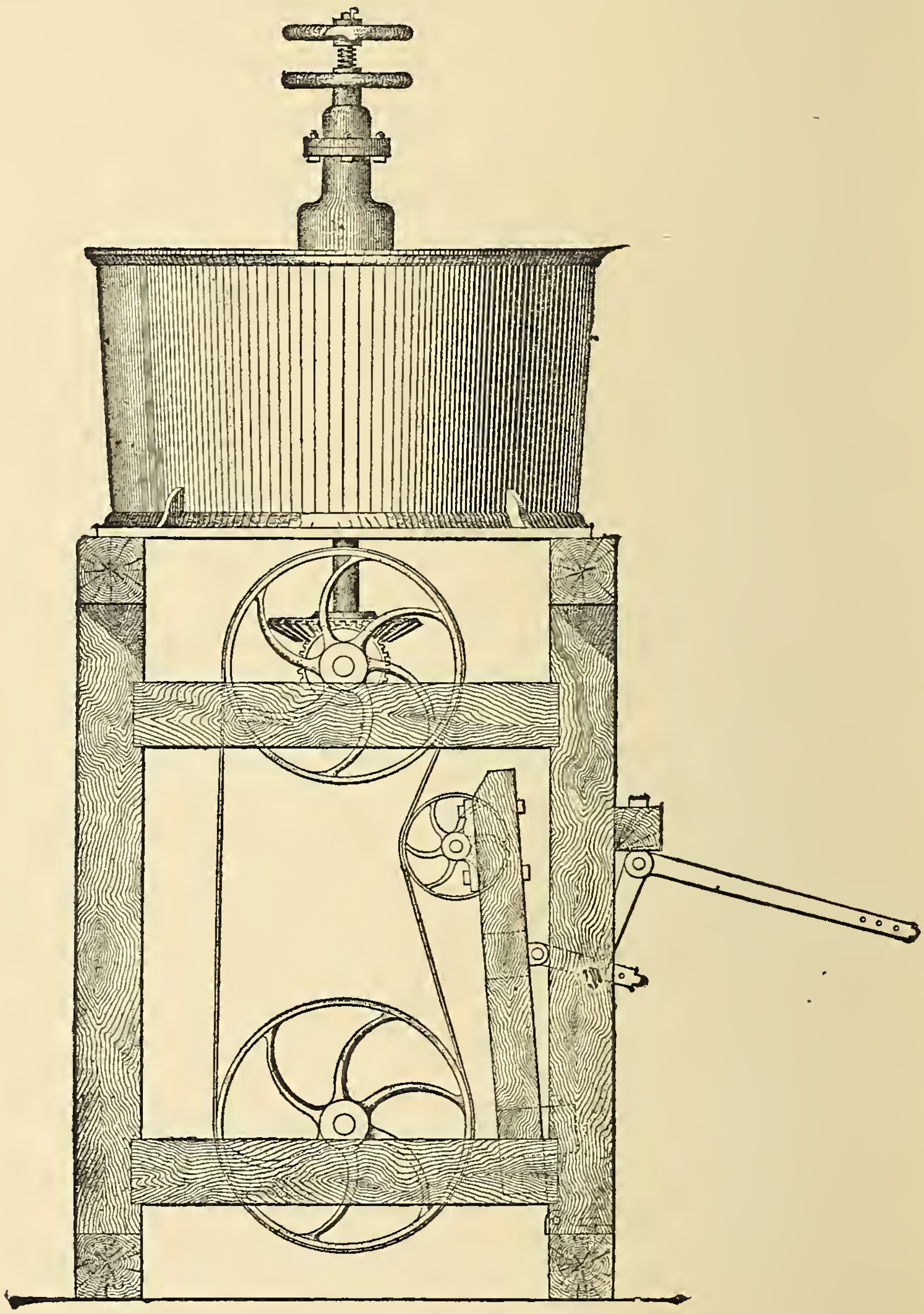
Scale $\frac{1}{48}$.

FIG. 21.—WHEELER AND RANDALL'S PAN.

Wheeler and Randall's Pan, also known as the *Excelsior*, is shown in Fig. 21. This pan differs from those before described chiefly in the form of the bottom, which is conoidal. The object of this device is to produce surfaces of such form as to insure perfect uniformity of wear and the highest degree of grinding effect. Its efficiency, in this respect, is attested by the experience of practical mill-men. It is not, however, so generally used as the ordinary Wheeler or other pans already mentioned.

The dies, muller, and shoes have, of course, a form corresponding to that of the pan bottom. They are secured in place in much the same way as in the Wheeler pan. There are guide plates to assist in directing the movement of the pulp, and

there are openings in the muller between the shoes for its free passage between the grinding surfaces. The gearing of the pan, step box, and driving shaft, and means of raising the muller,



Scale $\frac{1}{40}$.

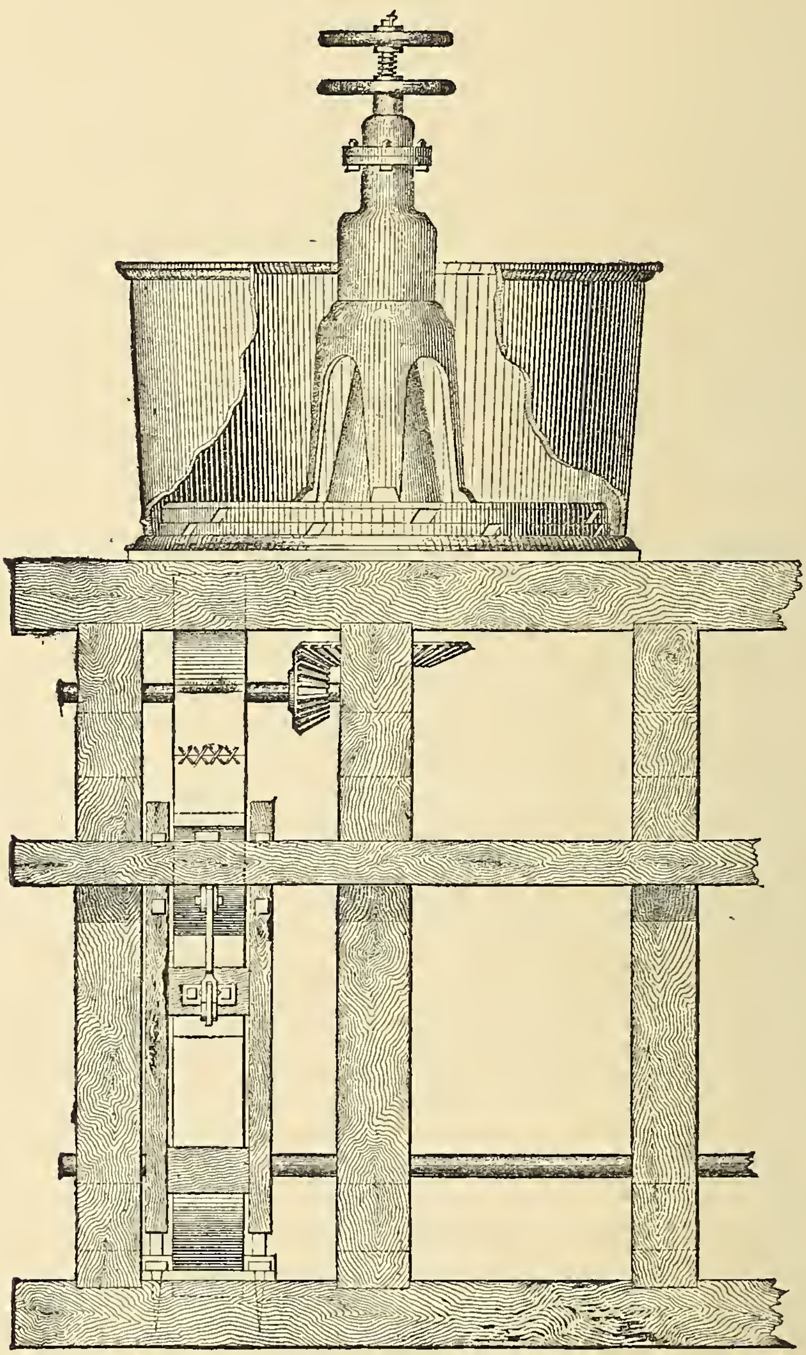
Fig. 22.—McCone's Pan. Elevation.

do not differ materially from the common Wheeler pan. This pan is made of various sizes; the largest is $4\frac{1}{2}$ ft. in diameter, and treats 3,000 lbs. of ore at a single charge. It weighs 5,000 lbs.

McCone's Pan.—The McCone pan, constructed by Mr. McCone, proprietor of the Nevada Foundry, at Silver City, is a large pan. Some of the details of its construction, and the method of setting it up, are shown in Figs. 22, 23, and 24. Figs. 22 and 23 show the pan as it is mounted on a timber framework, and the gearing by which it is set in operation. In Fig. 23 a portion of the pan rim is removed to show the interior. Fig. 24 shows a vertical section, and Fig. 25 a plan of the pan. In the latter a portion of the muller plate is shown and another portion is removed, exposing the shoes and dies below. This pan is 5 ft. in diameter and 28 in. deep, it is flat-bottomed and made either with or without a steam chamber. When the latter is desired, the false bottom is cast separately, with a rim an inch deep, and is then bolted to the main pan bottom, thus forming the chamber. There are no standards or legs for the pan to stand upon, the bottom being a square cornered plate of iron, projecting beyond the pan rim, and it may be bolted directly to the timbers on which it is to rest. The bottom, with its central hollow cone, may be cast in one piece with the pan rim, or, instead of the latter, a simple flange may be cast corresponding in size with the rim, to which flange the rim, which may be either a cast piece or made of sheets of iron riveted together, is bolted.

An improvement has lately been made to save the wear of the rim or side of the pan, and prolong its usefulness, by placing in the bottom of the pan a false rim or circular facing for the pan side, about 9 in. deep. This is cast in segments and made to correspond in form to the rim of the pan. When fixed in place it saves the pan rim from wear in that part which would otherwise suffer the greatest degree of friction, just as the shoes and dies protect the pan bottom and muller plate. When worn thin by the friction of the pulp the plates may be removed and new ones substituted for them. The driving shaft or spindle, *c*, passes up from below through the central hollow cone, *b*, but its point of support is usually independent of the pan, resting in such case in a step box, *h*, which is fixed on a timber below. Some, however, prefer to have hangers bolted to the bottom of

the pan and furnishing the support for the driving shaft, so that if the foundations of the pan settle, the relative position of the several parts is more readily maintained.



Scale $\frac{1}{40}$.

FIG. 23.—McCONE'S PAN. With portion of Rim removed.

The step box is cast in one piece, with a bearing for the end of the shaft on which the vertical mitre wheel and pulley of the common driving gear are fixed.

The driver, or hub, E, which is secured to the vertical shaft, is in two parts, an upper and lower. The upper is fixed to the shaft by two strong feathers or sliding keys, *k*. The base of the upper driver is cast with lugs, or projections, which fit into corresponding recesses in the top of the lower driver, by which means the latter is supported and set in motion. Above the upper driver is a cap piece, *j*, carrying the usual screw and nut arrangement for raising and lowering the muller, the bottom of the screw resting on the upper end of the vertical shaft. The lower part of the driver has three or four stout lugs or projections at its base, which fit into carriers on the circular part of the muller at *d*, Fig. 25. These carriers are also made to serve as the means of aiding the circulation of the pulp, as they assist in directing the current toward the centre when the muller is revolving.

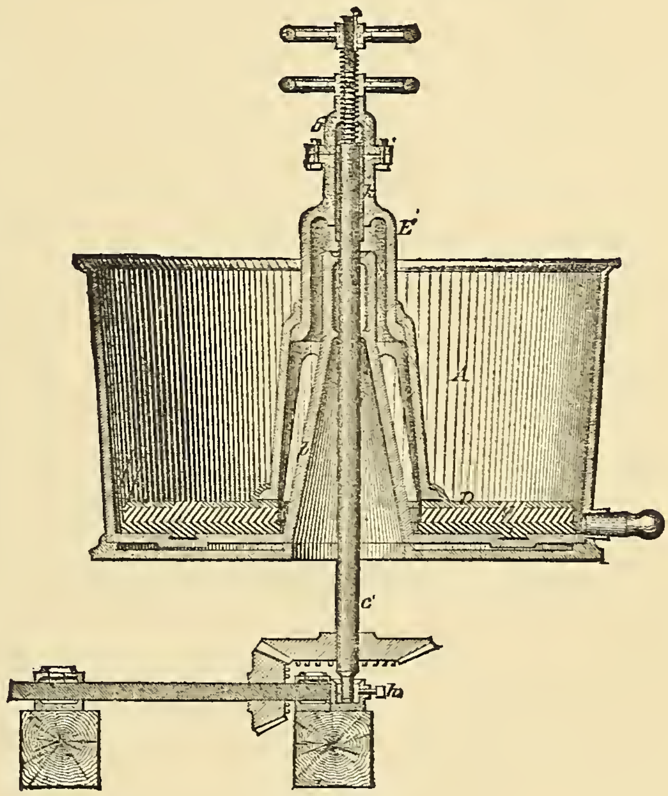


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FIG. 24.—McCONE'S PAN. Section.

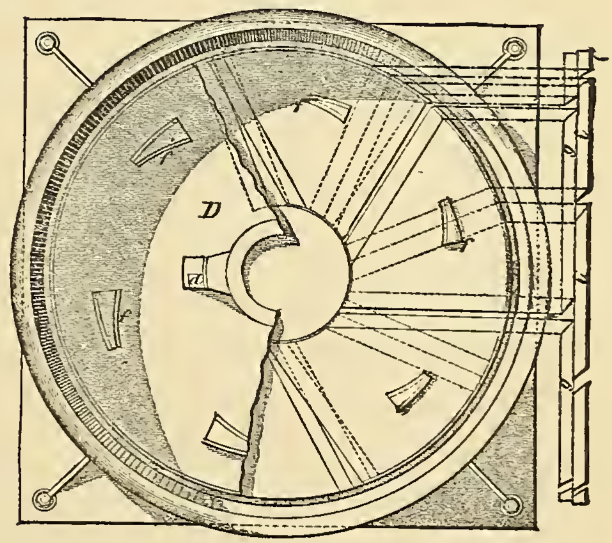

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FIG. 25.—McCONE'S PAN. Bottom, showing Shoes and Dies.

For this purpose they are sometimes cast 5 or 6 in. high, presenting a curved surface (not shown in the case illus-

trated) to the pulp, and forcing it toward the centre of the pan. By this means the guide plates, or wings, usually fixed to the side of the pan, but which to some extent obstruct the motion of the pulp, are dispensed with. Grooves for attaching guide plates are, however, cast in the pan rim, so that those who prefer may use them.

The dies and the shoes which are used in this form of pan resemble in many respects those of other pans. There is an inch and a half space between the outer edge of the die and the edge of the pan, and a similar space between the adjacent edges of the dies. The shoes, between which there are similar spaces, and which also have radial channels, or grooves, on their under side to facilitate circulation, have the same radial width as the dies. The radial width of the muller plate is a little less than that of the shoe and die, in order to allow a freer inlet and outlet to the pulp. The muller makes from 60 to 80 revolutions per minute. The pan takes 4,500 lbs. of pulp at an ordinary charge, and sometimes more. It is set up very simply, being bolted to timber supports below, and is put in motion or arrested by the application or withdrawal of a tightener to the driving belt, as shown in Figs. 22 and 23.

The Combination Pan (shown in Fig. 26) is a combination of the Wheeler and Patton pans, with a number of improvements upon both. Its capacity is 15 tons in twenty-four hours. It is very extensively used on the Pacific coast. When treating ores wet, the sides and bottom are cast in one piece, or the sides are made of wrought iron and riveted to a projecting flange on the cast-iron bottom. This mode of construction saves freight when it is a question of transportation to distant countries. When the pan is used for roasted ores the sides of the pan are of wood. The wings which project into the pan are made of copper plates, which assist the amalgamation. The muller is not cast on to the muller stem, but fits into slots, and when it is required to be loosened the motion of the muller has to be reversed; it can then be easily prized out with an iron bar. The muller is fixed in the stem by

means of a key in the screw thread; by taking out the key and turning the muller on the stem it may be raised and kept at any height for any length of time, and as easily lowered. The wings are attached to pieces of iron, which are dovetailed to fit into slots in the side of the pan.

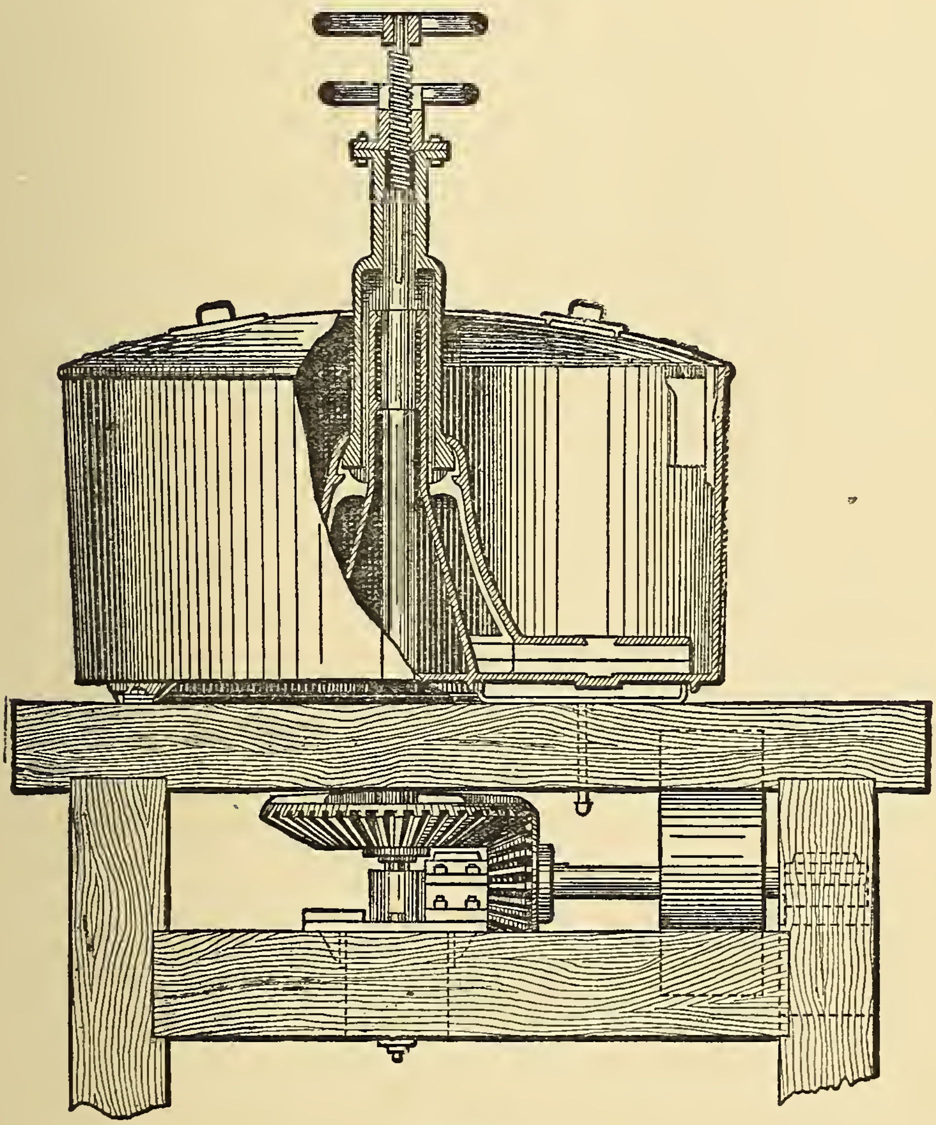

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Fig. 26.—Combination Pan.

Fountain's Pan and Horn Pan.—These resemble one another very much. The first-named has 5 ft. in diameter at the top, and $4\frac{1}{2}$ ft. at the bottom. The bottom, rim, and central hollow pillar are cast in one piece. A steam chamber, when desired, is provided by bolting to the bottom of the pan a circular plate that is cast with a rim an inch deep. The

upper edge of the rim is grooved out and a round piece of rubber packing laid in the groove, which, fitting closely against the pan bottom, makes a steam-tight joint and allows for the unequal expansion and contraction of the metal. The dies are attached to the pan bottom by means of a wedge-shaped projection, as described in the case of Greeley's pan.

The driver is cast in one long piece; its upper part, which is attached to the driving shaft by means of a key or feather, is cylindrical, and furnished on the inside with a long babbited bearing for the shaft; its lower part consists of three legs, or standards, each of which has a square lug or projection at the bottom, which, fitting into a raised clutch on the muller plate, carries and gives motion to the latter. The space between the legs or standards being open, affords free passage for the circulation of the pulp about the centre. In front of each lug on the standards of the driver is an iron plate with a flaring or irregularly concave surface, which when the driver is in motion tends to force the pulp to the centre; while directly in front of this contrivance—that is, in the direction of revolution—a large piece is cut out of the muller plate, thereby affording free passage to the pulp downward and between the grinding surfaces of the shoes and dies. Wings and guide plates are thus dispensed with, though the pan rim is cast with the ordinary means of attaching such plates if desired.

As the bottom of the pan is flat and the muller has the plane-circular form, the wearing effect on the grinding surface is much greater near the circumference than near the centre, owing to the difference in radial velocity. It frequently results from this that the shoes and dies of ordinary plane-circular grinding surfaces wear down much more rapidly at the circumference, leaving the metal thicker near the centre, and so producing an uneven bearing of the muller upon the bottom, and consequently an irregular movement.

In the Fountain pan the radial spaces between the shoes and between the dies are made wider, horizontally, near the centre than they are near the circumference, so that the area of grinding surface at the circumference may be more largely in excess

of that at the centre than it would be if those spaces were of uniform width, thereby obviating, at least in part, the inequality of wear. These spaces in some of the Fountain pans are filled with wood, as already described in pans of other makers. The shoes at their circumferential edges are provided with ploughs to stir up the quicksilver lying on the pan bottom.

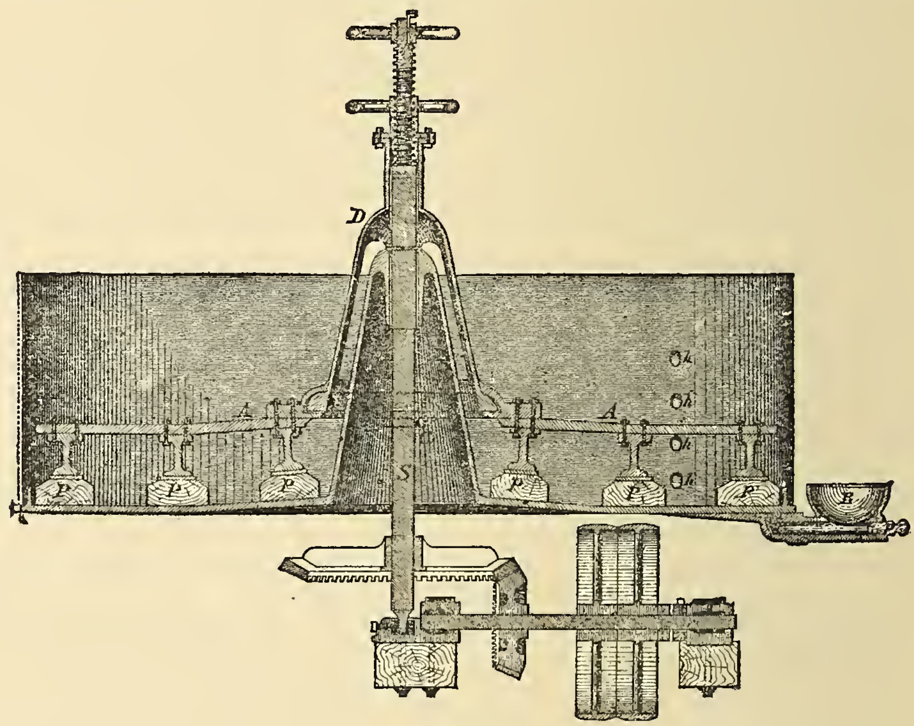
These pans work 3,000 or 4,000 lbs. of sand at a single charge. Their average duty, in working tailings, is stated at 10 tons per day.

Pans of much larger dimensions are used and have found great favour among mill-men. They treat charges of ore three or four times as large as that of the ordinary pans in the same, or but comparatively little more, time, economising thereby not only time but labour and power. One large pan requires much less machinery and fewer auxiliary parts in its operation than three or four smaller ones of equal capacity in the aggregate. The attention of the workmen is more concentrated and there is a much smaller loss, proportionately, by wastage of ore, quicksilver, and other material. While the time allowed for amalgamation is much less in the larger charge than in the smaller one, in proportion to the quantity of ore treated, the results, so far, seem to be nearly or quite as good. These considerations are of special importance in the working of low grade ores and tailings, which can be done profitably on a large scale and at small expense per ton, and in which the loss of a small percentage of the value is comparatively trifling in amount.

Settlers or Separators.—Their mechanical construction is as follows. A hollow pillar or cone, C (Figs. 27 and 28), is cast in the centre of the bottom, within which is an upright shaft, S, which is caused to revolve by gearing round the pan. To its upper end is attached a yoke or driver, D, which gives revolving motion to arms, A, extending from the centre to the circumference of the vessel. The arms carry a number of ploughs or stirrers, of various devices, usually terminating in blocks of hard wood, P, that rest lightly on the bottom. No grinding is

required in the operation, but a gentle stirring or agitation of the pulp is desired in order to facilitate the settling of the amalgam and the quicksilver. The stirring apparatus, or muller, makes about fifteen revolutions per minute.

A discharge hole, near the top of the settler, permits the water carrying the lighter portion of the pulp to run off, and at successive intervals the point of discharge is lowered by withdrawing the plugs from a series of similar holes, *h h*, in the side of the settler, one below the other, so that finally the entire



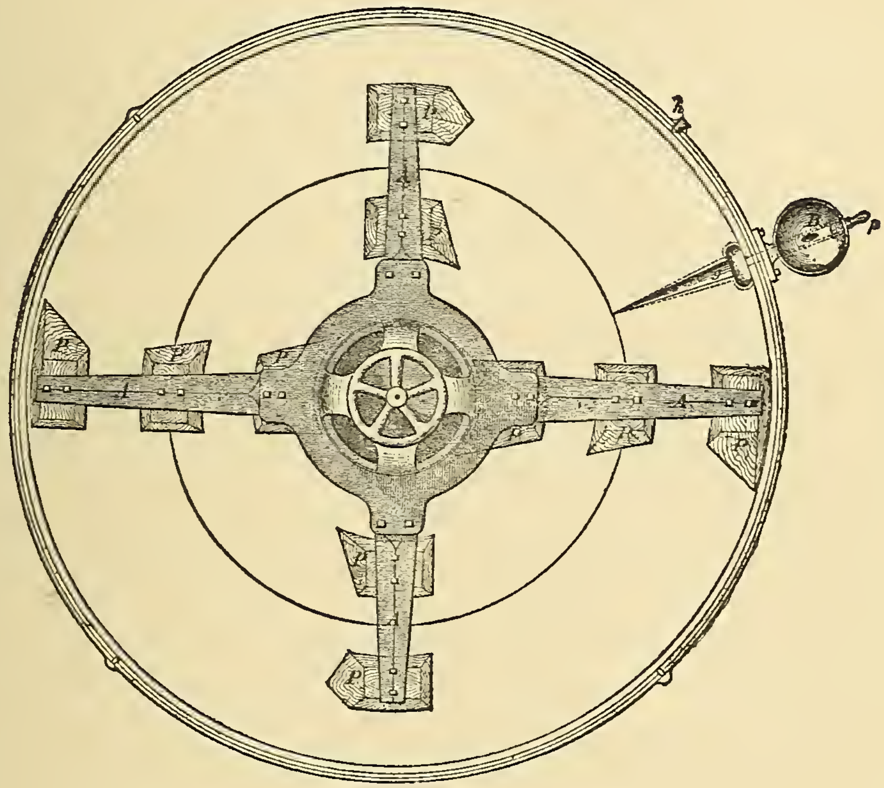
Scale $\frac{1}{48}$.

Fig. 27.—Separator or Settler.

mass is drawn off, leaving nothing in the settler but the quicksilver and amalgam. There are various devices for discharging these. Usually there is a groove or canal in the bottom of the vessel, as shown in Figs. 27 and 28, leading to a bowl, B, from which the fluid amalgam may be dipped or allowed to run out by withdrawing the plug, *p*, from the outlet pipe.

The settlers or separators, in which the quicksilver and amalgam are allowed to settle or separate from the pulp, after treatment in the pan, do not present so much difference in detail of construction as the pans do. They are made larger

than the pans, usually 7 ft. or 8 ft. in diameter, and are commonly made with a flat, sometimes concave, circular, cast-iron bottom, having a hollow cone or pillar at the centre, and a flange at the circumference, to which the rim, either of wood or sheet iron, is attached. The central shaft, with its driving gear below, the screw and nut arrangement above, for raising and lowering the stirrers, and the yoke or driver fitted to the revolving shaft, are not essentially different from the



Scale $\frac{1}{48}$.

Fig. 28.—Plan of Separator.

similar parts of the pans. Hangers are sometimes bolted to the bottom of the settler to carry the step box of the vertical shaft and support the driving gear; or these may rest on a timber frame independent of the bottom, as shown in Fig. 27. To the yoke or driver are attached four radial arms reaching to the circumference of the vessel. On each arm are two or sometimes three legs, terminating in a wooden shoe, variously shaped, which touches the bottom. These legs are movable radially, so that any one may be fixed at such point between

the centre and circumference as may be desired, and they are usually arranged at different distances on the several arms, so that in the course of each revolution each part of the surface of the bottom is passed over by one or another of the shoes.

The discharge of the separator is usually effected through outlet holes, already described, as shown in Fig. 27.

The quicksilver charged with amalgam is carefully cleaned by washing with water and removing from the surface the associated impurities, such as heavy particles of dirt, pyrites, &c.

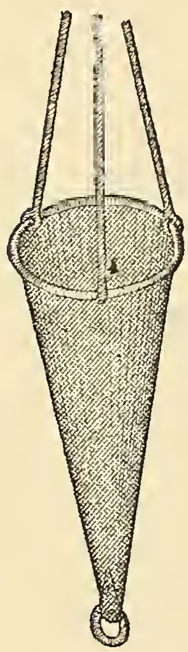


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FIG. 29.—AMALGAM STRAINER.

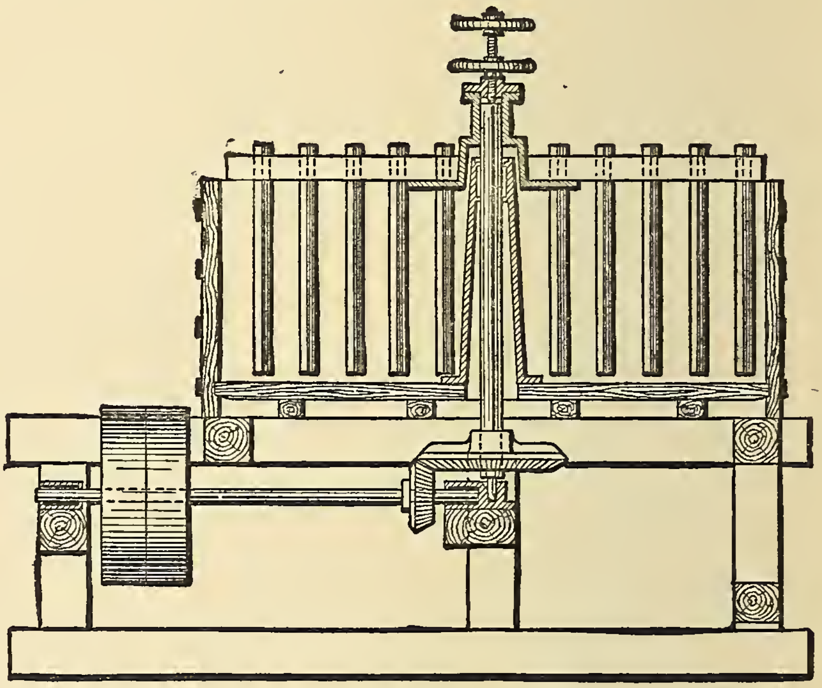

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FIG. 30.—AGITATOR.

In some cases the cleaning is performed in a small iron pan, resembling the settler in manner of construction, but much smaller, in which it is stirred slowly with plenty of clean water, which serves to wash out the impurities and remove them from the pan. When properly cleaned the amalgam is strained through a canvas filter or conical bag, 10 or 12 in. in diameter at the top, and 2 or 3 ft. long (see Fig. 29). The quicksilver is drained off and returned to the pans for further use, while the amalgam is thus obtained for the retort.

The method of withdrawing the fluid amalgam or quick-

silver from the vessel has already been indicated, and an ordinary contrivance for this purpose is shown in Fig. 28. Different makers vary this plan in some of the details. In some separators the groove or canal for the collection of the quicksilver is circular, concentric with the pan bottom, and usually placed midway between the circumference of the bottom and the base of the central cone. The outlet pipe for the discharge of the quicksilver and amalgam is connected with the bottom of this groove, leading out under the vessel to a point beyond the circumference, where it may terminate in a bowl, or may turn upward and be fitted with a vertical pipe in which the outlet may be fixed at any desired height, and the quicksilver allowed to discharge itself continuously as fast as it accumulates in the groove or receptacle in the pan bottom. A cock or plug at the lowest point of the discharge pipe permits the whole of the quicksilver to be withdrawn when desired.

Agitators.—The agitators (see Fig. 30), through which the pulp passes after leaving the separators, are in general wooden tubs, that vary in size from 6 ft. to 12 ft. in diameter, and 2 ft. to 6 ft. in depth. The main object in letting the stream of pulp pass through them is to retain and collect as much as possible of the quicksilver and amalgam and heavy particles of undecomposed ore that are carried out with the pulp discharged from the separator. A simple stirring apparatus, somewhat resembling that of the separator, keeps the material in a state of gentle agitation, the revolving shaft carrying four arms, to which a number of staves are attached. In some mills there are several agitators, in most cases only one, and by some they are not used at all. The stuff that accumulates on the bottom is shovelled out from time to time, usually at intervals of three or four days, and worked over in pans. Beyond these are a number of contrivances for concentrating the most valuable portions of the tailings. Among them are blanket sluices and other variously devised machines, some of which will receive further description later on, in the chapters on Concentration and concentrating machinery.

Clean-up Pans.—Fig. 31 shows a sectional view of a clean-up pan. In these the amalgam from the silver mill when dirty and impure is worked with additional quicksilver, and the waste matter washed off before retorting. Figure 31 shows the Knox Clean-up Pan, 4 ft. in diameter; wooden shoes are attached to the arms, and they are adjusted by means of the hand wheels on top of the driving spindle, to bear on the bottom of the pan, or not, as desired, the motion being communicated through the bevel gear underneath to the spindle.

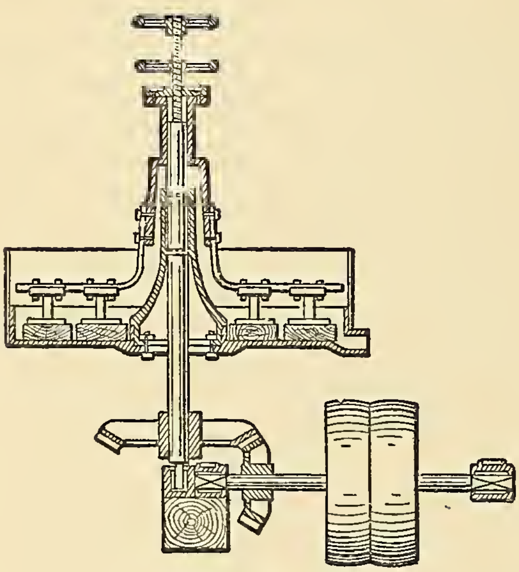

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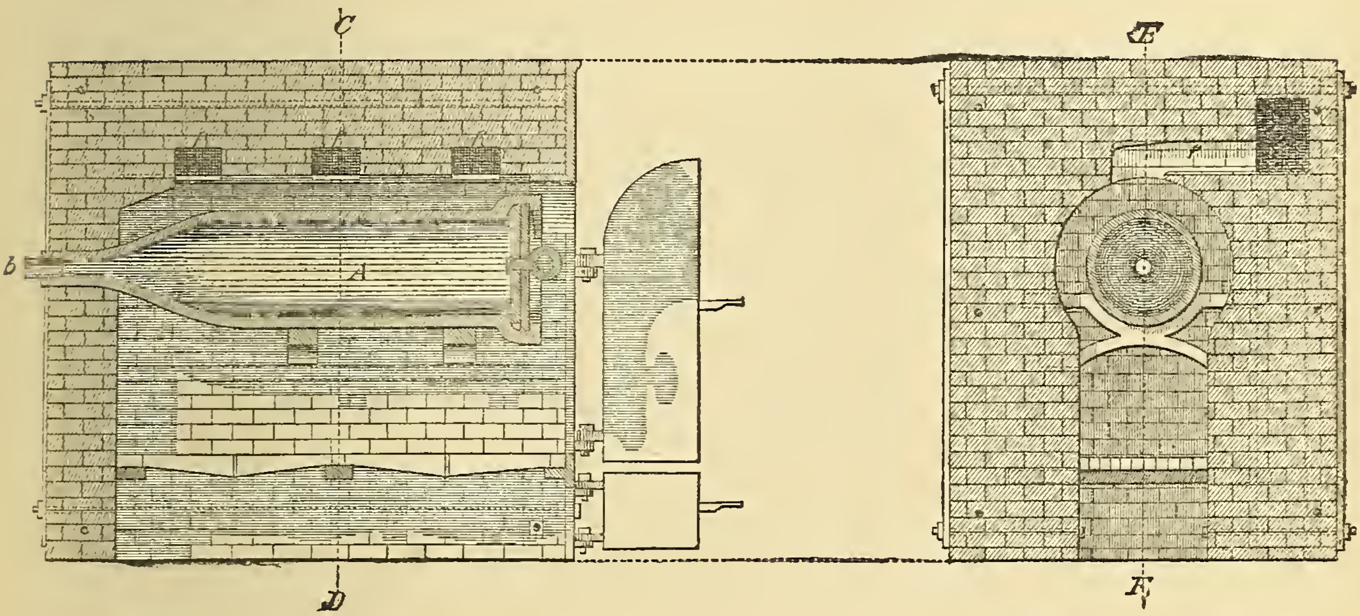
FIG. 31.—CLEAN-UP PAN.

Retorts.—The retorts in use are of various forms, but the most approved is cylindrical, about 12 in. in diameter inside, and from 3 to 5 ft. long; the casting being $1\frac{1}{2}$ in. thick (Figs. 32 and 33). The open end of the retort, or the end by which the charge is introduced, is cast with a flange or hood into which the door or cover of the retort is fitted. Within the cylinder portion of this flange or hood are two inclined lugs, opposite each other. A bar or bale turning upon a pin in the centre of the door holds it firmly in its place by the ends of the bar being turned under the inclined lugs. When charged, the joint between the door and the bottom of the flange is made tight by means of clay luting.

Another method of securing the door is shown in Fig. 34. The opposite end of the retort is usually made conoidal in form. In such cases the main cylindrical portion, 12 in. in diameter, is 3 ft. long, the diameter of the conoidal neck being thereafter gradually diminished to $2\frac{1}{2}$ in. at the extreme end of the retort, from which the exhaust pipe, *b*—the purpose of which is to afford escape to the volatilized quicksilver—turns downward and passes through the condenser which is shown

in Fig. 35. This is usually arranged on the principle of the Liebig condenser, and consists of a pipe, *a*, of considerably larger diameter than the exhaust pipe, *b*, so that the latter may pass entirely through the former, which, when in use, is kept constantly supplied with cold water by a pipe, *d*, opening into the bottom, the heated water flowing off at the outlet, *e*, near the top. The quicksilver, condensing in the exhaust pipe, falls into a receiver, placed under the end of the pipe, and which also is nearly full of water. The end of the exhaust pipe dips below the surface of the water to prevent access of air, but not suffi-



Scale $\frac{1}{45}$. Scale $\frac{1}{45}$

FIG. 32.—LONGITUDINAL SECTION OF RETORT. FIG. 33.—RETORT.

ciently to permit the passage of the water into the heated retort under any circumstances.

The retort is set in a brick furnace of simple construction, sometimes supported by a brick arch, through which a number of flues permit the passage of the heat from the fireplace below, sometimes resting on cross bars of iron, the ends of which are fixed in the brick sides of the furnace, as shown in Fig. 32.

Directly below the retort, extending under its whole length is the fireplace and ash-pit. Above it is an arch, from the top of which the flues, *f*, lead to the stack. Some retorts are set in such manner that temporary brickwork may be built up in front

of the door during the sublimation to prevent the escape of heat. Dampers are so arranged that the heat may be applied more or less vigorously to the front, back, or middle of the retort, according to its requirements.

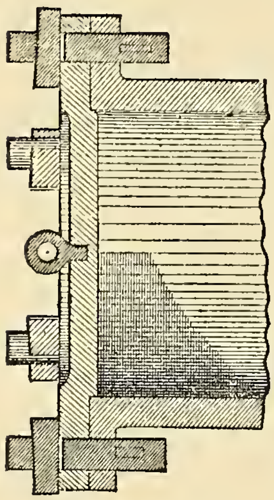


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FIG. 34.
RETORT LID.

The retort is furnished with amalgam trays (illustrated by Fig. 36), having a semicircular shape adapted to the bottom of the cylinder. There are usually several of these, and they are of a size convenient for handling when loaded with amalgam. In many mills, however, they are not used, the amalgam being charged directly upon the bottom of the retort. Before introducing the amalgam into the tray or the retort, the surface of the latter is covered with a thin wash of clay or slime, such as is produced in stamping, to prevent the metal from adhering to the iron. Whiting, wood ashes, or paper are sometimes used for this purpose, and recommended as being less likely to choke the pores of the bullion. The amalgam being placed in the retort and the door properly adjusted and luted with clay, the fire is lighted and heat is applied, at first very gently and afterward gradually increased. If heated too strongly at first, the surface of the bullion in contact with the retort is liable to fuse and prevent the escape of quicksilver from the central part.

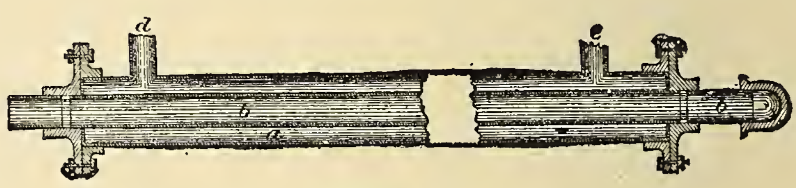

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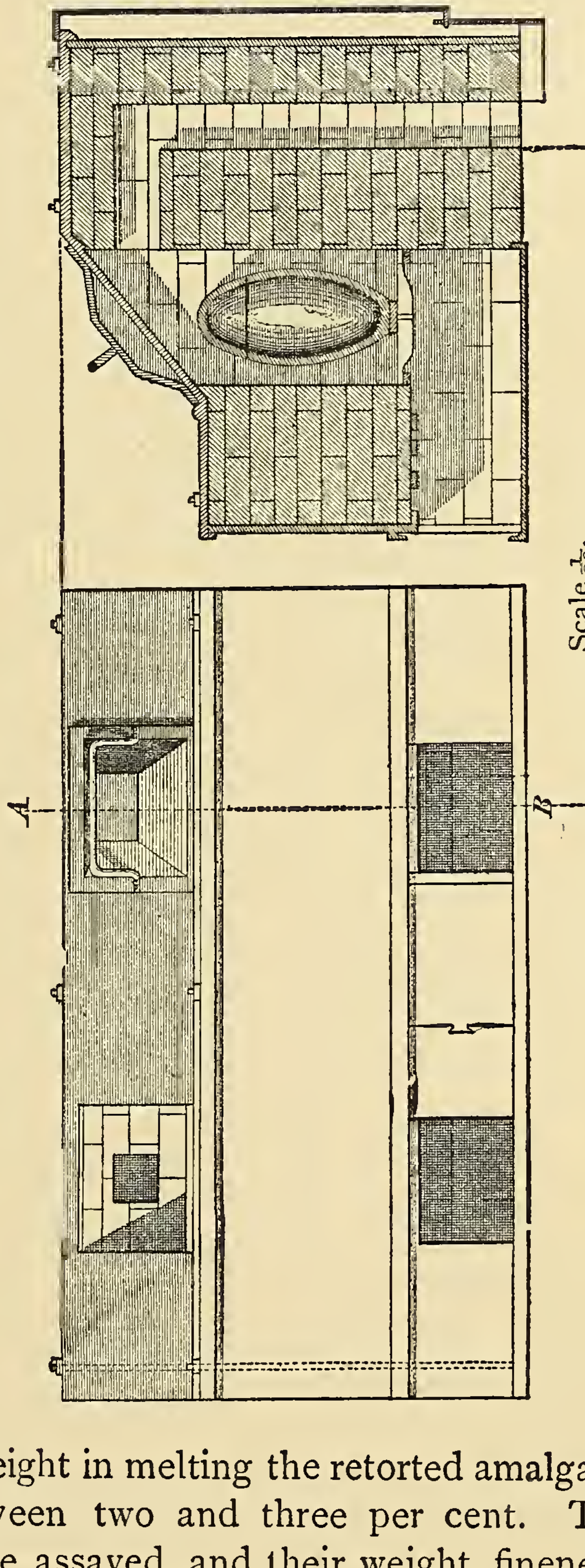
FIG. 35.—CONDENSER.

Scale $\frac{1}{22}$.

FIG. 36.—AMALGAM TRAYS.

The charge for a cylinder of the dimensions above described is about 1,200 lbs. The firing usually occupies about eight hours. When quicksilver ceases to volatilize, the retort is

gradually cooled down and the bullion withdrawn. About one-sixth of the original charge usually remains, or 200 lbs. of crude bullion from 1,200 lbs. of amalgam. The trays have generally a partition in the centre, and if the retorting has been properly conducted the chunks of bullion are ordinarily of convenient size for handling, but when spongy and voluminous they are put in an iron box to be broken up with a hammer in sizes ready for the melting-pots, and then cast into ingots.

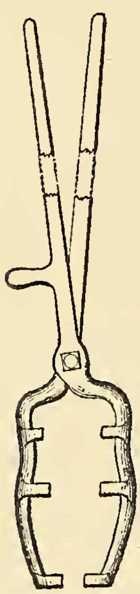


Scale $\frac{1}{30}$.
FIGS. 37, 38.—BULLION MELTING FURNACES.

Melting Furnace.—The form of furnace commonly used for this purpose is shown in Figs. 37 and 38. Fig. 37 is a front elevation of a double furnace. The cover of the left hand furnace is removed, showing the size and position of the flue. Fig. 38 is a transverse section on the line A B in Fig. 37; Fig. 39 shows the tongs for removing the melting-pots. Fig. 40 shows the form of ingot mould usually employed. The loss of weight in melting the retorted amalgam, or crude bullion, is between two and three per cent. The ingots, when obtained, are assayed, and their weight, fineness,

expressing the proportions of gold and silver contained in thousandths, as well as their coin value, are stamped upon them.

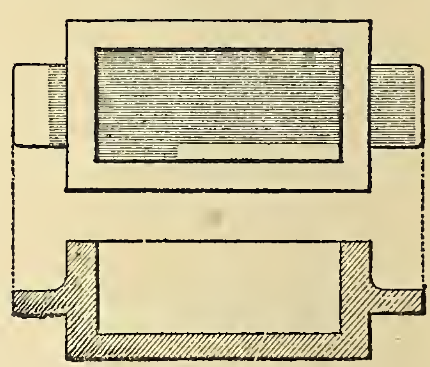
General Arrangement of Mills.—The general arrangement of the machinery in a mill, working silver ores by the method described in the foregoing pages, may be illustrated by a drawing in Fig. 41, which presents a sectional view of the building and the more important machines employed in it.



Scale $\frac{1}{27}$.

FIG. 39. BULLION MELTING TONGS.

The batteries of stamps, as many as there may be, are arranged in one straight line. Behind them, that is on the feed side, is the breaking floor, where the rock is prepared by a stone-breaking machine, or, in its absence, by hand. When the slope of the ground permits it, large bins are sometimes constructed above and behind the breaker, into which receivers the wagons or cars bringing the ore from the mine may discharge their contents. As the outlet of the bins is on a higher level than the mouth of a breaker, the rock is delivered to that machine without much handling. Such bins, where practicable, are of great advantage in providing a reserve of ore for the mill whenever communication with the mine is interrupted for a time.



Scale $\frac{1}{15}$.

FIG. 40.—INGOT MOULDS.

From the batteries the crushed ore is discharged upon an apron—or, as in the case illustrated, into a trough or launder—which conveys it to the settling tanks. These stand directly in front of the batteries, though in some mills, for lack of space, they extend along the adjacent side of the building. A platform is usually provided upon which the pulp may be deposited when shovelled out of the tanks. Some mills are so arranged as to use a car, in which the pulp is moved from the tanks to

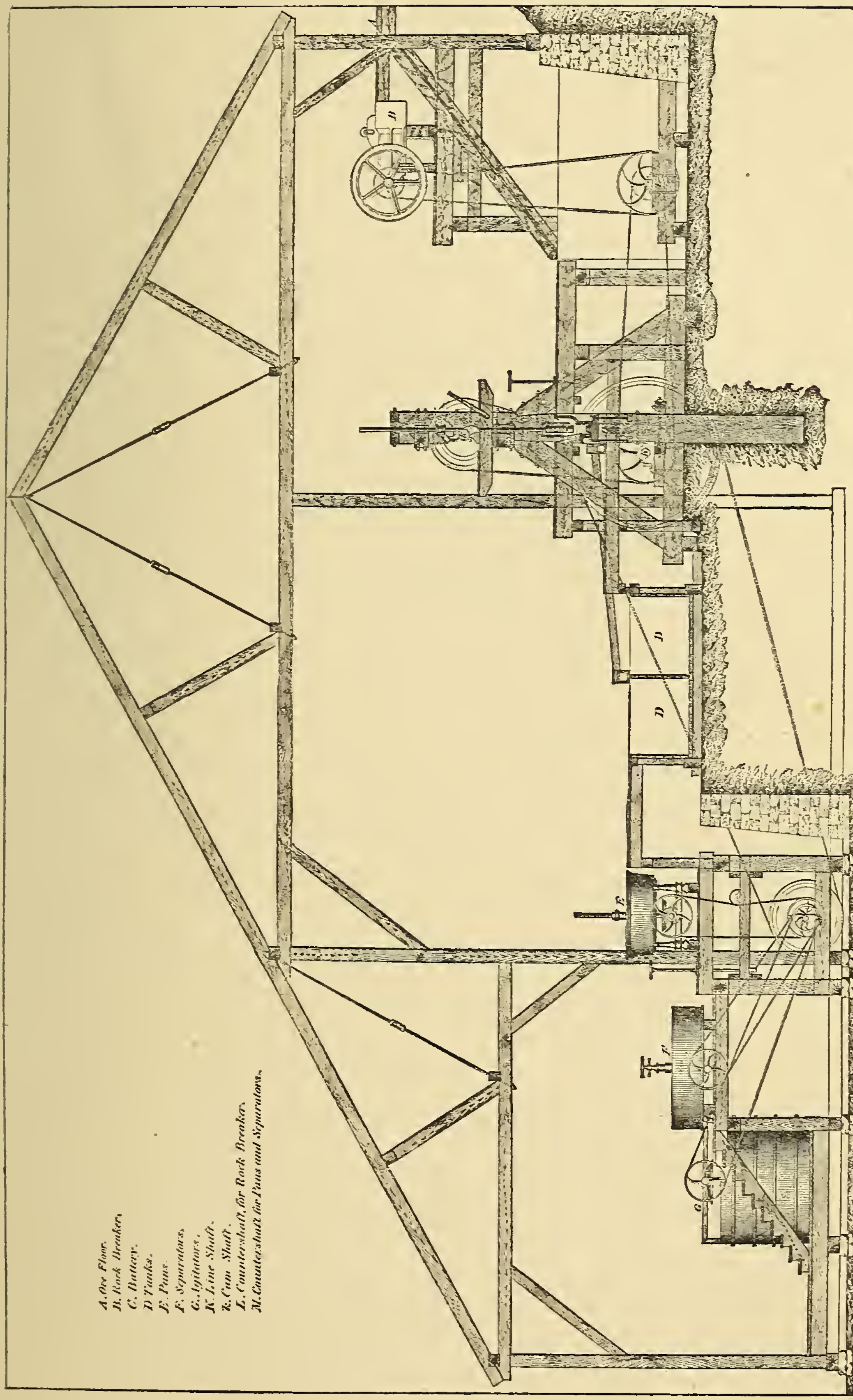


FIG. 41.—SECTION OF SILVER MILL FOR WET CRUSHING. Scale $\frac{1}{200}$.

the pans. This is especially necessary when the tanks are more remote from the pans, or when the latter are arranged in a line at a right angle to the line of the batteries.

Generally the pans are arranged in a straight line, parallel to the line of batteries, as in the case illustrated. The separators stand in front of the pans, arranged in a parallel line, and on a sufficiently lower level to permit the charge of the pan to run into them. Below the separators are the agitators, or other similar contrivances, for the purpose of preventing the escape of quicksilver or amalgam.

The power is usually communicated from the steam-engine——or whatever motor is used in the mill—by gearing or belting to a line shaft, which is placed in front of and parallel with the line of batteries. On this shaft are pulleys, opposite to those of the several cam shafts, to which they transmit by belting the power necessary for the stamps. The same shaft imparts motion, by means of countershafting and belting, to the rock-breaker and to the pans. For the latter a line of shafting is usually arranged under the row of pans, from which shaft each pan, separator, agitator, or other similar machine, may be driven by a separate pulley. The power required for each stamp of ordinary or average weight, with due allowance for friction, is about 1½ horse-power per stamp. The power demanded by the pans is from 3 to 6 horse-power, according to their size and capacity. The expenditure of power per ton of ore crushed, ground, and amalgamated—judging by the relation existing between the power of the engines provided and the work performed by the mills—is between 1½ to 3 horse-power, averaging about 2, but varying according to the capacity of the mill and the economy with which the power is applied.

CHAPTER IV.

COST AND RESULTS OF MILLING OPERATIONS.

Cost of Labour and Materials—Staff required in Mill—Cost of Treatment per ton of Ore—Mills on the Comstock Lode—Sampling the Ore—Yield of the Mills—Cost and Results at the California Mill—Loss of Quicksilver in Milling—A Pan Room Illustrated.

Cost of Labour and Materials.—The number of men employed in a well-managed mill of twenty-five stamps and ten Greeley or large Wheeler pans, having a total capacity for treatment of about 50 or 55 tons of ore per day of twenty-four hours, is as follows :—

2 breaking rock and supplying the feeders—both by day;
2 feeding the batteries, if no self-feeder used—both by day;
3 tankmen, discharging the tanks and supplying the pans—each working eight hours;
2 amalgamators, 2 helpers, 2 engineers—one each by day and one each by night;
1 foreman, and 1 mechanic. In all 15 men.

The price of labour varies from 12s. to 24s. per day—averaging, perhaps, 14s. per day for the several classes employed. The actual cost of labour per ton of rock treated would, however, reach a somewhat higher figure in the course of a year than is indicated by the foregoing list of employés, owing to unavoidable loss of time for repairs or other hindrances, which diminish the actual capacity of the mill. The average cost of labour per ton of ore is from 6s. to 9s.

The other chief elements of cost are iron, consumed in wear of castings for stamps and pans, averaging about 2s. per ton; quicksilver consumed or lost in the amalgamating process, of which the amount is rarely less than 1 lb., and frequently 1½ lb. per ton, costing about 3s.; fuel, the cost of which varies

from £1 to £3 per cord of wood, according to the distance of the mill from sources of supply, and varying therefore from 4s. to 12s. per ton; water, when purchased, about 1s. 3d. to 1s. 8d. per ton of rock; other materials and incidental expenses making in the aggregate an important item; and, finally, transportation of the ore from the mine to the mill, varying from 4s. to 16s. per ton, which, if not properly an item of milling expense, enters into account as an offset to cheap fuel or water power, when these can only be had at a great distance from the mines. A 50-stamp silver wet crushing mill requires about 14 to 15 cords of wood daily to run it.

The items of cost for treating a ton of ore can be distributed as follows:—

	£	s.	d.
For labour	0	9	0
,, wood, $\frac{1}{6}$th of 1 cord . . .	0	8	0
,, quicksilver, $1\frac{4}{5}$th pounds . .	0	3	0
,, castings, $6\frac{1}{2}$ pounds . . .	0	2	6
,, sulphate of copper, $1\frac{3}{4}$ pounds .	0	1	0
,, oil	0	0	6
,, hauling and sundries . . .	0	10	0
Total per ton	£1	14	0

This cost has been much diminished of late years, and ore has been treated in the year 1879 at a cost of 24s., exclusive of transportation. Where water power is employed, ore can be treated at below £1 per ton; and in one district, after railway facilities were given the cost was reduced to 16s., but so low a rate is attainable only in mills of large capacity.

Mills on the Comstock Lode.—The mills working on the Comstock ores are located at various distances from the mines, some being in the immediate vicinity, while the most remote are from thirteen to fourteen miles away.

Nearly all the mills of the district are "custom mills," and the greater part of the ore produced from the various mines is worked in such. "Custom mills" are those that receive the ore from the producer, work it at a fixed price per ton for treatment, and return to the customer a certain percentage of the value of

the ore, the latter having been previously determined by assays. The ores are carefully sampled, both before delivery to the mill and after crushing at the mill, as will be explained in more detail below, and a return of 65 per cent. of the assay value is required of the millowner. Falling short of this in bullion, the mill is bound to make good the deficiency, while any excess obtained by so much of the process of treatment as has been already described belongs also to the mine or customer.

Some mills make returns as high as 70 per cent., or even more, but (as might naturally be expected) the average return of the mill to the mine does not exceed the requirement. The residue, or "tailings," of the ore, after it has passed through the separators, belongs to the mill, and this is in many cases made to yield a good result on re-working. It is therefore the interest of the millowner, when working at a fixed price per ton, to treat as many tons as possible, and to return no more bullion in excess of the required standard than may be necessary in order to maintain a good reputation among competing mills. Sixty-five per cent., accordingly, does not fairly indicate the value actually extracted from the ore, and is not a fair criterion by which the efficiency of the process may be judged.

Sampling the Ore.—As the mill is required to return to the mine a certain percentage of the value contained, it is necessary to have the ore carefully sampled and assayed beforehand in order to get at a basis of settlement. This cannot be accomplished without some trouble and expense, a large number of assays being necessary in order to obtain a reliable average result. Therefore assays are made of a double set of samples of every lot of ore. One set of these samples is taken from the wagon taking the ore from the mine to the mill: the other is taken from the mill after the ore has been crushed.

The wagon sample is obtained by drawing from every wagon load of ore dispatched to any mill a sample of the rock, keeping distinctly separated from each other the samples of ore sent to different mills. Each of these samples is assayed, or sometimes a number of samples of loads that were all sent to the same

mill during one day are mixed together and the assay taken from the mixture, the mill being charged with the weight of ore and the amount of gold and silver represented by the assay.

The mill sample, which is to serve as a check on the wagon sample, is usually taken by allowing the crushed ore as it comes from the battery to run into a pail or other suitable vessel, which is held or placed at the end of the trough or apron leading from the batteries to the tanks. A sample is taken in most mills every hour, in some every half-hour, and accumulated samples taken during a single day are well mixed together and dried. From this mixture a sufficient quantity is drawn for assay. Where there are several batteries, running on different classes of ore, each battery is sampled separately.

The wagon samples and mill samples always differ somewhat in yield, and the former is usually the higher in value. It is reasonable to believe that the finely crushed material being intimately mixed by the process should furnish a fairer average, nevertheless the value of the mill sample depends very much on the manner in which it is taken. Not only may it chance that the sample is caught just after an unusually rich or poor shovelful of ore has been supplied to the battery, but accident or neglect of proper precaution may affect the value of the assay. The pail or vessel placed to catch the crushed ore should not be allowed to become more than half full, and no water should be permitted to run over, as a sort of concentration would take place immediately. It may happen that the trough or launder is placed unevenly, so that more water runs off at one side than at the other, and unless the sample represents the whole stream its value is likely to be greater or less according to the point where it was taken.

In some mills the samples are taken from the ore tanks after the sand has deposited itself in them, either collected on the surface or drawn out from a considerable depth by means of a tryer or tube. Not only may accident determine the value of such a sample, but in the sand of the tanks the value of the slimes that have passed on without depositing themselves, and which are often quite rich, is not represented.

While making two sets of samples, the mine reserves the right to settle according to the wagon sample, but in practice both assays are duly considered and an adjustment arrived at.

Yield of the Mills.—The impression that only 65 per cent. of the value is obtained by pan process, and that 35 per cent. is lost is erroneous, for the return of 65 per cent. is based on the result of treating the ore in the pan and collecting the amalgam in the settler. In some mills the additional product of the agitator is returned with that of the pan and settler, while in other mills this is not done, especially if the required standard of 65 per cent. has been already reached by pan and settler without further addition. Moreover, the return of 65 per cent. includes nothing of what is or may be obtained from the subsequent treatment of slimes and tailings; and furthermore, it is to be considered that the ore, as charged to the account of the mill, contains an average of 6 or 7 per cent. of moisture, for which in the return no allowance is made; the sample for assay, by which the return is made, being previously dried, 65 per cent. of the dry sample is really equivalent to 69 or 70 per cent. of the wet rock.

Cost and Results at the California Mill.—The California Mill, situate at Virginia City, in Nevada, is one of the finest in the world, and is well managed; it contains eighty stamps which fall 7 to 8 in., and make ninety to one hundred drops per minute. The ores are very easily crushed, and the full capacity of the mill is estimated at 360 tons per twenty-four hours, or 4½ tons per head of stamps per day; probably the average work is 320 tons a day, or 4 tons per stamp; the stamping is done wet. The stamps are fed automatically, and the total number of men employed in the stamp mill per twenty-four hours is twelve, namely, two engineers, four feeders, two firemen, two repairers, and two bin-men.

From the stamps the ore runs through a trough to the pan mill, situated at some distance down the cañon. In this there are forty pans running on ore and four pans on tailings. The

average charge of pans is 3,600 lbs. of ore worked in five hours; the mullers run about ninety revolutions, and bear on the bottom or grind. From 12 to 24 lbs. of salt and about one-third as much bluestone is added to the charge, and after three hours' grinding 350 lbs. of mercury is added. These quantities vary with the richness and character of the ore. The average loss in mercury is from 2 to 2½ lbs. per ton of ore. The settlers (twenty) run about eighteen revolutions and two and a half hours, and their tailings run into the agitators and thence over the blanket sluices. The staff in the pan mill per twenty-four hours includes six amalgamators, twenty tank-shovellers, two pan-shoers, two amalgam-men, two engineers, two firemen, five woodmen, three repairers, three oilers, one retorter, one extra roust-about, one lamp-boy, two night watchmen, two foremen, one superintendent; total, fifty-four men.

The tailings form about 10 per cent. of the ore, and are re-worked in the tailing mill, and then run over blanket sluices. This requires two amalgamators, four shovellers, one cart-man, seven blanket-men, and one boss blanket-man = fifteen men, or a total in stamp, pan, and tailing mills, of eighty-one men per twenty-four hours. The wages are from $3 to $4 per day.

The fuel consumed per twenty-four hours in the battery mill (360 tons) is 80 to 90 cords, and in the pan and tailing mills (369 tons) 38 to 40 cords; total, 120 to 130 cords, at $10 per cord.

There are two sets of blanket sluices, each of six tables 300 ft. long, with a grade of about 2½ degrees. In the first set ⅝ of all the tailings recovered is caught, and ⅜ in the second set. About 8 per cent. of the stamped product is saved in these sulphurets, which assay sometimes $20. These blanket tailings are mixed with from 3 to 10 lbs. of salt per ton, exposed to the air, and then mixed with the tailings of the settlers in the agitators, and the mixture is stated to average only about $7 per ton.*

* I publish the figures in this account of the California Mill as supplied to me; but in several particulars—notably the consumption of fuel and the yield and assay of the tailings—the figures may require explanation.

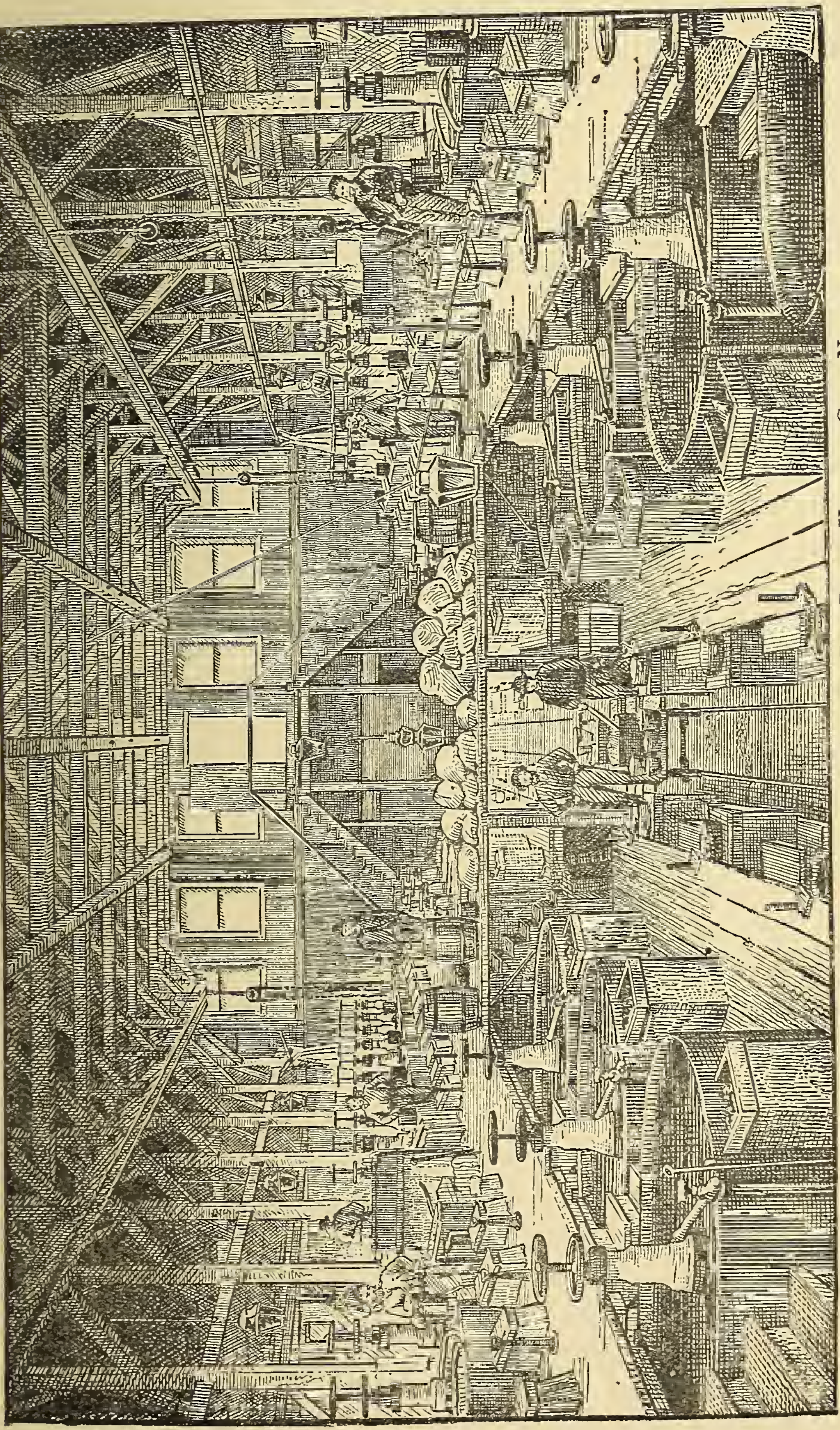


FIG. 42.—VIEW OF THE CONSOLIDATED VIRGINIA PAN ROOM, VIRGINIA CITY, NEVADA.

Loss of Quicksilver in Milling.—The loss of quicksilver is always considerable in a mill, no matter how clean or careful the men may be about their work. Some way or another it gets all over the mill in small particles, even on the top of the roof, where it is carried by escaping steam.

Quicksilver charged with copper readily becomes coated with small particles of iron. In the pulp it is readily coated by iron pyrites, grease, slimes, &c., or reduced to great fineness by grinding. In these floured and coated conditions much of it will float away and be lost, unless means are employed to collect it. Cyanide of potassium is an excellent cleaning agent. Ores containing much talc likewise act unfavourably on quicksilver. As soon as quicksilver is fouled, and becomes sluggish, it not only loses to a large extent its amalgamating power, but is also easily cut up and floured.

In addition to the sources of mechanical loss, much of the quicksilver is lost also chemically. The water from the settlers, if filtered and concentrated, will show quicksilver present in solution. Sulphate of copper in solution is decomposed by quicksilver, some of the quicksilver becoming sulphate of mercury, while the precipitated copper forms a copper amalgam with the remaining quicksilver.

Chloride of silver also can be decomposed by quicksilver, chloride of mercury being formed. If binoxide of manganese is present in the ore, it occasions a heavy loss of quicksilver.

It is of importance to keep the pan as clean as possible of quicksilver in the first half of the period of working the charge. I also consider it of importance to reduce the grinding time to a minimum, and to experiment how long an ore should be ground to give the best result before adopting a fixed system.

A Pan-room Illustrated.—Fig. 42 (p. 95) shows the pan-room of the Consolidated Virginia Mill, which (like the California Mill just referred to) is situated at Virginia City, Nevada.

CHAPTER V.

TREATMENT OF SLIMES AND TAILINGS.

Slimes and Tailings on the Comstock Lode—Amalgamation of Slimes in Pans—Concentration of Tailings—Tailing Reservoirs—Successful Concentration of Slimes and Tailings—At Tombstone—At the Montana Company's Mills—Amalgamation of Raw Tailings in the Washoe District.

Slimes and Tailings on the Comstock Lode.—The treatment of the residue, or that which remains of the ore after it has been subjected to the process already described, is a matter of much importance.

The term "slimes" applies to that portion of the crushed ore which is reduced by the stamps to an exceedingly fine condition, and, flowing from the batteries in the stream of running water, does not find sufficient opportunity to deposit itself in the tanks in which the coarser sands are collected, but is carried beyond them, and only settled, after a long time, either in another set of tanks or in large reservoirs. These are properly called "battery slimes," to distinguish them from the material that may be reduced to a similarly fine condition by the operation of the pan, as already mentioned on page 33.

The term "tailings" is understood especially to apply to that portion of the crushed ore which, after having been subjected to the grinding and amalgamating process in the pan and settler, flows away from the latter or from the agitator, and passes on out of the mill deprived of the greater part of its valuable contents. A part of this material is in a very fine and slimy condition, but the bulk of it may be better described as a fine-grained sand. Leaving the mill the stream flows onward

and is usually subjected at once to various methods of concentration, the most common of which is the blanket table, by which means a portion of the escaping amalgam, quicksilver, and heavier particles of ore may be extracted to be re-worked, while the great mass of material is finally collected in dams or reservoirs for still further treatment. Reservoirs for this purpose are placed at convenient points along the courses of the streams, or cañons, on which the mills are usually placed, though on account of the limited space in the narrow valleys they are necessarily small; but at the mouth of the cañons they are of larger capacity. The quantity of slimes produced in crushing ore varies considerably in different mills. In some mills the proportion of slimes is thought to be about 2 per cent. of the ore crushed by the stamps, while in others it is said to be as high as 10 per cent. This is partly due to the difference in the character of the ore or its gangue, and partly to the difference in the conditions under which the crushing takes place.

As these slimes carry with them much of the very finely crushed silver ore, their assay value is not only considerably higher than that of common tailings, but is often higher than that of the original ore. Especially that portion of the silver-bearing mineral of the ore which exists in the form of rich sulphurets, being soft and readily crushed, is liable to be reduced to an impalpably fine condition, particularly if freed from particles of quartz that might if present preserve it in a coarser form and escape with it from the battery before being reduced to slime.

Amalgamation of Slimes in Pans.—The attempts made to work slimes by ordinary methods in pans at first gave unsatisfactory results. This has been attributed partly to the finely divided and clayey condition of the material itself, the particles of quicksilver and amalgam becoming coated with an adhering film of the slimy substance, preventing amalgamation and involving great mechanical loss of quicksilver; partly also to the probable existence of the silver in the form of sulphurets, as just indicated. Owing to the difficulties of working the slimes,

it has been the custom to mix a part of the slimes with the sand of the tanks, or in other cases with common tailings, and so work them over in pans. In other mills the stream bearing the slimes has been allowed to run off with the common tailings, finding its way to the grand reservoirs at the mouths of the cañons; while others have accumulated them in dams, made specially for that purpose, holding them in reserve for a time, when they may be turned to some account.

Much progress has been made in working the slimes in pans without previous roasting, and several mills of considerable capacity have been devoted exclusively to this business, purchasing their supply of slimes from neighbouring crushing mills, and treating them in such manner as to obtain a fair percentage of their value with considerable profit.

The method of treatment employed in working slimes in these mills does not differ much in mechanical details from that by which the fresh ores are worked, the most notable feature of the process being the use of much larger quantities of chemical reagents than is customary in milling ordinary ore. The reagents themselves do not differ in kind, but the quantity is increased to an extent which makes it possible to believe in their efficient action.

In the Janin Mill four McCone pans were in use, each of which received 2,500 lbs. of slime at a charge. The charge of ore for this pan is 4,000 or 5,000 lbs.; but as slimes increase greatly in bulk on the addition of water, it will not take more than 2,500 lbs. of the dry material.

Twelve pounds of the sulphate of copper and 36 lbs. of salt are put into the pan with each charge, and the whole is worked for two hours before putting in the quicksilver. Little or no grinding is required, as the material is already exceedingly fine; the muller is raised high enough above the bottom to avoid unnecessary friction, but it is revolved at about the same speed as in working ore, the main object being to keep up a rapid and perfect circulation of the pulp. After two hours the quicksilver is added, and in large quantity, usually 300 lbs. The charge is then worked for four hours longer, and afterward

drawn off into the settler, from which the amalgam is collected in the manner already described, while the residue is allowed to pass through large agitators before finding its way to the tailing stream, in order to save as much as possible of the escaping amalgam and quicksilver.

The quantity of quicksilver employed in this process is so large that the loss of that metal in the operation is proportionately great, especially as it is believed that the clayey condition of the slimes greatly facilitates its escape. This loss is stated at about 5 lbs. of quicksilver to the ton of slimes. This item, together with the cost of the chemicals—which, by reason of their liberal use, amounts to a considerable sum—makes the treatment of slimes quite expensive, probably not less than £2 10s. per ton.

The supply of slimes is obtained by purchase from the neighbouring crushing mills, their value being previously determined by assay. This value varies from £5 to £10 per ton, and the purchase price for some time past has been from 12s. to £1 or more per ton, according to the contents. It is said that the method of working just described extracts upwards of 60, and frequently 80 per cent. of the assay value.

In Mr. Parke's mill, in the same neighbourhood, the method of operation is very similar to that just described. He used, however, large wooden pans or tubs, having cast-iron bottoms but wooden sides. The tubs have large capacity. The wooden sides of the tub, which are 3 in. thick, are furnished on the inside with a lining, also of wood, 1 in. thick, which, when worn down, may be replaced by new pieces without reconstructing the tub. In addition to this inner lining it has been found advantageous to attach strips of wood, 2 by 4 in. thick, to the inner circumference of the tub; these strips stand vertically, and about 2 in. apart, producing a rough or corrugated, instead of a smooth, surface on the inside of the tub. This has been found to assist greatly in the disintegration of the lumps of slime, which, although consisting of the most minute particles, hold together like clay, with great tenacity when wet, and in ordinary pans frequently present a serious obstacle to

thorough amalgamation. Mr. Parke has carried this improvement still further by making this corrugated surface on a rim of cast iron. The corrugations may be on the rim which is cast on the pan bottom for the purpose of attaching the wooden staves of the side, or the rim may be cast separately and placed in the pan. The corrugated surface is 10 in. high and the rim is 3 in. thick. As the wear of a pan rim is confined chiefly to within 10 in. of the bottom, the provision of this surface, which, like the shoes and dies, may be easily renewed when worn down without changing the pan, is deemed a great improvement.

The O'Hara and Thompson roasting furnace was also introduced for the purpose of roasting the slimes with salt, and then submitting them to amalgamation.

Concentration of Tailings.—Tailings in the Washoe district have generally been found more profitable than slimes. It has been already said that the stream of water carrying the tailings out of the mill is usually passed over blanket tables, in order to save all that can possibly be obtained in that way.

The blanket table, the most common means of concentration, is a long shallow trough, about 20 in. wide, with sides only 1 or 2 in. high, and of indefinite length, according to the supply of tailings, water, the character of the ground, and other conditions. A number of these tables are usually established side by side, sometimes only two, three, or four together, sometimes as many as fifteen or twenty. They are inclined gently, usually having a fall of 6 or 12 in. in 12 ft. of length. They are covered with coarse blankets made especially for the purpose, in strips about 2 ft. wide, and cut in such length, usually 10 or 15 ft., as may be deemed convenient for removal and washing. As the stream of tailings runs over the blankets the heavier portions of the ore, sulphurets, &c., and particles of amalgam, are retained in the blankets, while the poorer sand is washed away. The quantity of water must be carefully adapted to the purpose, sufficient to prevent the accumulation of sand and not enough to carry away the heavier particles. The operation is usually assisted by a man who, with a broom, sweeps

the surface of the table lightly, aiding the even distribution of the material and exposing the particles more thoroughly to the action of the water. The blankets are taken up from time to time and washed out in a tub of water, usually once in twelve hours. While the blankets of one table are being washed, the stream is turned so as to run over the neighbouring tables. The concentrations washed from the blankets are collected and worked in pans. They usually yield from £3 10s. or £4 to £6 per ton.

The profit accruing from this source to the mill reduces considerably the original cost of crushing and amalgamating.

Various other contrivances for concentration have also been introduced.

Tailing Reservoirs.—The tailings coming down the cañons from the mills above, after having passed over the blanket-tables, or having been subjected to other methods of concentration, are finally allowed to accumulate in reservoirs. Some of these, of small capacity, are placed along the course of the streams, but the principal deposits of that sort are on the level land adjacent to the mouths of the cañons. Thus at Dayton, where Gold Hill Cañon opens upon the plain, there are two or three reservoirs, the aggregate contents of which probably amount, at present, to 400,000 tons. This quantity is daily increased by what is brought down from the mills above. Further down the river, near the mouth of Six Mile Cañon, and receiving everything brought down from the mills on that watercourse, is another known as the Carson reservoir, containing not less than 200,000 tons of tailings. In Six Mile Cañon, two miles above its mouth, is a smaller reservoir, formerly estimated to contain 100,000 tons, but of which a large portion was swept away some years ago by freshets. The quality of the tailings in these dams varies considerably, depending on several conditions; among others, the proportion of slimes that may be mixed with the sands. Thus assays of the slimy and richer parts may show a value of £5 or £6 per ton, while the coarse sands vary in value from £1 to £2 or £3

per ton, according to the original character of the ore and the degree of efficiency with which its valuable contents have been extracted. The smaller reservoirs around Dayton are generally richer than the large ones.

Since the discovery of the Consolidated Virginia and California ore bodies on the Comstock lode—generally known as the famous Bonanza mines—immense quantities of tailings have been accumulated which are now being treated.

Successful Concentration of Slimes.—In later years considerable attention has been devoted to the question of concentrating the tailings on proper machines. This is by no means an easy question when it is considered that after the fine pulverization of the ore in the battery the same undergoes a grinding process during the pan amalgamation, lasting from three to four hours, whereby the ore is reduced into an almost impalpable powder. As for the slimes, they consist generally of the finest particles of the chlorides, and the easily friable brittle silver ores, float gold if any, and the clay or talcose portion of the ore. As the name "slime" indicates, it is a slimy, pasty agglomeration of matter, which one would hardly judge capable of bearing a mechanical separation; but it has been found that with the best of concentrators, like the Frue vanner, and properly constructed buddles, a proper concentration of tailings and slimes may be effected. They have been thus successfully dealt with at several mills, and as examples I may cite the cases which follow.

Concentration of Tailings at Tombstone, Arizona.—The ores from Tombstone mine above water-level were mostly chlorides in quartz gangue, containing also lead carbonate, manganese and iron oxides, and some sulphides of silver, iron, copper, lead, and zinc. They assayed on an average 60 oz. of silver and $\frac{1}{5}$ oz. of gold per ton and 3 per cent. of lead. With depth the chlorides gave way to the less tractable sulphides. Tellurides of silver and gold also made their appearance. The surface ores were closely milled, averaging 85 per

cent. of the silver and 45 per cent. of the gold contents. These results disappeared as the ore lost its chloride character and began to carry its silver in the form of sulphide.

Up to March, 1884, the mill had amalgamated 89,608 tons of ore, estimated to contain 4,168,527 oz. of silver and 18,244 oz. of gold, and had produced 3,225,110 oz. of silver and 9,454 oz. of gold. This would leave tailings containing about 10·5 ozs. of silver and 0·098 oz. of gold per ton, or a value of $15·60 at the legal rate for silver. The actual value at market prices at the time of the operation was about $12 per ton.

Mr. T. A. Church, the superintendent of the mine, gives the following description of the apparatus and machinery which he erected for the concentration of the tailings :—

"The amount of silver and gold locked up in tailings, which were accumulating at the rate of 15,000 to 25,000 tons a year, and were worth about $12 a ton, was so great that its recovery became a serious problem. The ore being thoroughly oxidized, no benefit could be expected from weathering, and amalgamation having failed once to extract the metals in the tailings, it was assumed that it would fail again. Roasting and chlorination were prohibitive by their cost, which was not less than $20 or £4 a ton at that time and place, or twice the value of the tailings. Guided by the fact that the lead carbonate carried a high percentage of silver, experiments were immediately made in concentration. A second amalgamation was also tried and yielded about $1·50 per ton.

"At that day, 1883 to 1884, the experience had in concentrating pan tailings was not at all reassuring. Experiments had been made, but without success, except on ore that contained as much as 8 or 10 per cent. of lead. Ores of 3 per cent. lead, like those in Tombstone, had not been successfully treated. The outlook for utilisation of the concentrates was also poor. No iron ore for flux was at hand, and the only substance that was free from silica was limestone. There were on the company's property some mines of manganese ore, carrying about 20 oz. of silver to the ton. This ore would not amalgamate

well, and trials made subsequently proved that it would not concentrate well.

"Under these circumstances it was determined to undertake concentration of the tailings on a large scale, and if this succeeded to attempt to smelt the concentrates in a shaft furnace with manganese as the flux.

"Experiment showed that the tailings could be concentrated either on the Frue vanner or on the German rotating round table. The principal difficulties experienced were from the extreme fineness of the tailings, which had been stamped through screens of 30 and 40 mesh, giving pulp of $\frac{1}{40}$ to $\frac{1}{60}$ in. diameter as a maximum, and from this down to the finest particle. The proportion of very fine slime was increased greatly by grinding in the pans, so that probably 60 per cent. of the ore was in the condition of slimes sufficiently fine to form a tenacious mass, though no more than 3 or 4 per cent. of clay was present. In the course of the work two mills were built.

"*First Concentrating Mill.*—The mill first built contained six Frue vanners and three round tables, the latter serving entirely as tailing machines to the vanners. An ordinary agitator, such as is found in pan mills, 9 ft. in diameter, served to receive the dry tailings taken from the old beds and mix them with water. This mixer was connected with a mill of 15 stamps, the tailings from which ran directly to the mixer. The three discharge-plugs, usual to settlers, were retained, but were bored with 1-in. holes, limiting the discharge, so that the pulp was kept at a certain height in the mixer. With about nine revolutions per minute of the mixer arms, the current was not sufficient to carry off the heaviest part of the pulp. This settled to the bottom of the mixer, and was shovelled out periodically. It was rich enough in lead for smelting, and was richer than any other part of the product in gold and silver.

"The rest of the tailings ran from the mixer to the six Frue vanners, on which it was distributed without sizing. The tailings from the vanners were collected in a belt elevator, and raised high enough to run on the round tables.

"The vanners worked well when the quantity passed over

them was restricted, but when it reached 5 tons a day the fine part of the slimes was carried over and lost. On this account the saving of lead in this mill was much less perfect than afterwards when facilities for handling larger quantities were obtained. This mill ran for about a year and a half. The work done in the first thirteen months is shown by the following table:—

THIRTEEN MONTHS' WORK AT FIRST MILL.

	Tons.	Assay per Ton.			Per Cent.
		Silver.	Gold.	Lead.	
		oz.	oz.	per cent.	
Mill-tailings treated . .	11,467	13·21	0·22	8·00	
Production:—					
From the mixer . .	93	52·65	0·48	23·20	
" " Frue vanners .	1,483	45·22	9·58*	30·90	
Percentage saved by weight.					13·31

Weight of tailings required to make one ton of concentrates, 7·5 tons.

Percentage saved, by value: Silver, 41·29; Gold, 34·09; Lead, 50·61.

"Although the tailings treated in this mill were estimated to contain 8 per cent. of lead, it was known that the old beds as a whole would not contain more than 3 per cent., the stock for this mill being taken from the best and richest parts of the beds.

"*Second Concentrating Mill.*—The attempt thus made to concentrate was not considered an entire success until the round tables were put in. They were introduced to prevent the loss of fine material which passed over the vanners, and they proved to be so well adapted to the treatment of the finest slimes that a new mill was designed in which they bore all the work of concentrating the slimes, while jigs were made to treat the coarse product that had settled in the mixer. In addition to the changes, sizing was introduced, both by trommel screens and by hoppers with a rising current of water. The old mixer was replaced by an agitator made like a pug mill. It consisted

* This is probably erroneous, as $7\frac{1}{2}$ tons of tailings, worth 0·22 oz. gold, could not produce 9·58 oz. in the concentrations.—M. E.

of an upright box, 30 in. square, with the corners filled in to make an octagon. A shaft making 105 revolutions per minute stands in the centre, and carries a frame or basket of $\frac{3}{4}$-in. iron bars, which rotates just within a number of rods that project inward from the sides of the agitator. The construction of this machine gave more trouble than any other part of the mill on account of breakages. At first the central shaft carried radial arms that ran between the fixed arms of the box; but when the latter were cut off and the former reduced to a basket of vertical arms running past the ends of the fixed rods, the machine ran for months without a stoppage.

"This mill was designed to use dry tailings from the old beds, and also the current product of tailings from a mill of 20 stamps, treating from 40 to 60 tons a day. The mill tailings were drawn from the settlers, each of which contained two pan charges, or about $2\frac{3}{4}$ to 3 tons of ore with about 1,200 gallons of water. The discharge of a settler took place in about five minutes, and to prevent a rush upon the concentrating machines the mill tailings were received in a box, about 16 ft. long and 2×2 ft. section, with three discharge openings at one end. When a settler was discharged this box would fill, and the heavy tailings settled to the bottom. As soon as the flush was over the surplus water would run out, and the deposit in the bottom of the box was removed gradually by a stream of clear water which poured with some force from a spout. All of the water needed for mixing with the dry tailings passed in this way through the equalizer.

"The equalizing box, as this contrivance was called, discharged into the agitator or mixer, which was fed with dry tailings by hand.

"The dry tailings were wheeled from the old settling pits to a platform, and then shovelled with regularity into the agitator. Shovelling stopped when the mill was sending down tailings, and was resumed when the flush began to decline. From the agitator the tailings, now become pulp, ran to a belt elevator with 12-in. buckets, lifting 26 ft. high, and entered the mill at one corner of the building. From this point they ran through

two trommel-screens to a series of six hoppers, forming the sizing apparatus and occupying the upper story of the mill. The last two of these hoppers were quite capacious, holding probably 600 gallons. They would fill to the brim during a flush, but in the intervening period the water level in them would sink so low that the discharge launder of the fifth hopper was not covered, and the sixth received no pulp. During a flush the water-level would rise in the hoppers, and their spouts discharged an increased quantity of pulp, that showed its effects immediately in a thicker deposit on all the concentrating machines. Thus the effects of flushing the tailings down from the mill were met partly by stopping the use of dry tailings, partly by storage in tanks and hoppers, and partly by greater supply of pulp to the concentrating machines, and it was constantly shown that all of these resources were needed to reduce the fluctuations due to this cause to their lowest possible value as disturbing factors.

"The trommels had punched screens of $\frac{3}{64}$-in. mesh, wire screens being used also, but giving trouble from the opening of the meshes. The trommel rosettes had round wrought-iron spokes, and a 1-in. blank nut was strung on each spoke to give percussive action. Both trommels sent their through-fall to the line of hoppers and their coarse stuff to the first jig. The second jig was supplied from the first hopper, and the second, third, and fourth hoppers supplied three round tables, while the fifth and sixth hoppers supplied three other round tables. The jigs had percussive action, the pistons being carried by springs, and forced down by the blows of a ram. The speed, 120 strokes per minute, was not enough for such fine material.

"The round tables were all 15 ft. in diameter, turning 105 times in one hundred minutes, and had a slope that varied from 7 in. in $7\frac{1}{2}$ ft. for coarse slimes, to $4\frac{1}{2}$ in. for fine slimes. All of them were covered with Akron cement, which is well adapted to this use. Brushes were not used, the ore being cleaned and the concentrates washed off by jets of water. These machines did excellent work. The cement surface, combined with the thinning of the stream of pulp as it spread from the centre to

the circumference, caused the retention of the fine carbonate of lead and other heavy minerals most perfectly. It was proved that when the concentrates were once deposited, the losses by cleaning were very small, most of the loss occurring in the very fine slime that ran over the table on the raw pulp. The mill was overcrowded, and with slower work it is quite certain that much better results could be obtained even upon pan tailings. Two jigs and six tables were expected to treat 120 tons a day, but they frequently treated 150 to 170 tons at a time when the tailings were so fine that the jigs did not do their proper share of the work. Some of the tables treated a ton an hour constantly.

"In considering the following table it is to be remembered also that the choicest part of the tailing beds had been removed before the second mill was built.

One Year's Work at the Two Mills, April 1st, 1883, to March 31st, 1884.

	Old Mill.	New Mill.	Total.
Days run	126	144	270
Tons treated	3,346	13,623	16,969
New tailings		6,150	6,150
Old tailings	3,346	7,317	10,663
Ore crushed		156	156
Product, tons	395·20	1,495	1,890·20
Ratio, tailings to product			1 : 8·9

Per cent. saved by concentrates	Gold	48·81
	Silver	40·57
	Lead	72·06
Do. by tailings	Gold	55·53
	Silver	53·11
	Lead	77·61

"The actual saving was greater than the table shows. The tailings from the tables on which the finest slimes were treated were run through a series of six settling tanks, and a product containing about 8 to 10 per cent. of lead and 12 to 15 oz. of silver obtained. This was used as a binding substance in making the concentrates into bricks, but no account was taken of this material in estimating the work of the mill.

"There were two constructive defects in the mill. The sorting hoppers proved to be very inefficient. It was not necessary to sort the stock closely, but it was very important to separate the finest slime from coarser sizes; and this the hopper failed to do. Every one of the concentrating machines that depended on the hoppers received a considerable proportion of slime; and it is likely that much of the richest and heaviest part of this slime followed the coarse table stock to those tables which had the greatest slope and were least suited to its treatment. The hopper shape seems to be the worst that can be utilized for the action of a rising stream. Many experiments were made to ascertain the source of the losses, and always with the result of locating them mostly in fine slime contained in the pulp after it had run over a table, that is to say in material that never rested on the table surface. When once deposited, the heavier portion of the pulp could be washed clean with very little loss.

"The other defect was remedied easily enough. The coarser grains of the tailings were rounded, and were apt to run over the table surface without stopping. The Frue vanners were replaced to work up this part of the stock, and their percussive action proved to be useful.

"It is impossible to say how far the losses could have been obviated by a more perfect system of sorting. Very much could have been done, but it is not probable that the losses would have been reduced below 25 per cent. except by crushing the coarse part of the tailings so as to unlock the minute particles of silver minerals enclosed in them. Probably one half the loss was due to this cause and one half to slimes.

"It was proved conclusively that concentrating machines can deal successfully with the finest slimes. A seventh round table was built for the purpose of reducing the silica in the extremely fine slimes, that were used for binding materials in making the concentrates into bricks for the subsequent smelting operation in the shaft furnace. This material was finer than flour, and was about like the dust that rises on a light wind. Even the heavy concentrates could be blown away by a light breath, though no trouble was experienced in saving them.

"The cost of the work varied with the constantly increasing distance of the tailing beds from the mill. The first mill was run by water power, and used a large proportion of tailings direct from the amalgamation, and under these circumstances the cost was about 92 cents per ton. When the tailings were brought in by hand, water power being retained, the cost rose to $1·39, both of these data relating to Frue vanners. When the tables were put in and a larger quantity of material handled and the treatment of tailings direct from amalgamation was resumed the cost was $1·23, though steam power was used exclusively, and increased the cost 24 cents a ton. These figures of cost cover the experimental stage of both mills, and should be reduced materially for regular and experienced work.

"The quality of the concentrates was excellent. They contained more than 50 per cent. lead, and an amount of silver that varied according to the material from which they were made."

Concentrating Tailings at the Montana Co.'s Mine.*— Here, as at Tombstone, pan tailings from one of the company's mills are carried by launders to the tailing mill, where they undergo a preliminary sizing in a series of spitzkasten, and discharge through a syphon on to the Frue vanners. Each of the spitzkasten—there are five of them—discharge each on to a separate vanner. The tailings leaving the vanners show that the work has been well performed, and that a large percentage of the mineral particles have been collected on the vanners; and what goes to the pit, although not worthless, assays very low. (The amount is not stated.) In the 50-stamp mill twenty Frue vanners are set up, so as to receive the discharge from the agitators and the pulp from the end of the vanners, or tailings flow to the tailing pit to undergo an "oxidation process," with a view to further treatment subsequently. The subjoined paragraphs are in the director's own words:—

* I am indebted to the Directors of the Montana Mining Company for the information here given. An account of the operation of the Montana Company's mills will be found later on, p. 125.

"*Tailings, Water Concentration.*—The tailings, which have been twice concentrated, are composed of sand of various grain-sizes and degrees of richness. The ore, chiefly sulphides of silver, antimony, and copper, is mostly attached to and enclosed (with a limited amount of gold) in these grains of quartz. In themselves, the grains will not readily concentrate so as to afford a product sufficiently valuable to bear the cost of carriage to the smelting works, its metallurgical treatment, and at the same time yield a satisfactory profit on the operation. Further, water concentration, from the character of the grains, would more or less result in destructive concentration; that is, in order to obtain a limited quantity of saleable ore, by far the greater proportion of bullion indicated by assay would still form part of the subsequent tailings. A second reduction of the sand to liberate the ore, even if it could be cheaply accomplished, would only increase the difficulties of water concentration, while the process itself would in all probability entail an amount of cost which the material could not bear. Experiments made on the pulp produced in the month of July, 1886, showed that forty per cent. of the total weight passed through a sieve mesh of 14,400 holes, and twelve per cent. through 22,500 holes to the square inch. Assuming, however, that concentration could be effected with reasonable success, the process would require (1) a copious supply of water over and above the supply to the mills, which does not exist; (2) perfect protection of the dressing apparatus during eight months of the year from the influence of a freezing temperature; and (3) a large extent of covered surface mechanically heated throughout the period referred to. The want of surplus water and prevalence of a temperature from freezing point to 40° below zero for two-thirds of the year, render the question of treating the tailings—which will be some 200 tons daily—by water dressing appliances more or less of an impracticable character; while, in other respects, if these conditions did not prevail, an hydraulic process may be regarded as one not likely to afford an adequate return on the capital outlay which

would be required, or to compensate for the skill, care, and attention which would be necessary.

"*Mill Treatment.*—Two well-known mill methods are open for the treatment of the tailings:—

"(1.) One involving drying, chloridizing and pan treatment with mercury.

"(2.) Storing the tailings in dams, leaving time, air, and chemical agency to effect their oxidization, and then treating them in pans with mercury.

"The first method admits of treating the tailings as produced, but it would necessitate the outlay of money for drying, chloridizing, and pan apparatus, and entail a daily cost for drying, chloridizing, and amalgamating the tailings. No reliable data have yet been obtained as to the amount of sulphur which may be present in the tailings, the proportion of salt required for chloridizing purposes, the percentage value of bullion obtainable, or the time necessary for conducting the amalgamating part of the operation.

"The second method is more simple:—By storing the tailings in dams and allowing a sufficient measure of time for decomposing and oxidizing the sulphides present, the ore is rendered fit for pan amalgamation. No drying or chloridizing would be required, and, as a matter of course, no apparatus or attendance on this division of the work would be wanted. But against this method there are some drawbacks: (1) it is not exactly known what time (whether two, three, or four years) will suffice for completing the oxidation of the ore; (2) the tailings must be stored in dams specially constructed for the purpose; and (3) the operation which in the first place could be conducted at the mines must, perhaps, be performed at a distance from the present mills. The apparent advantages and drawbacks of each of the two methods, failing the discovery of a more advantageous one, will have to be carefully weighed, but an approximate estimate of results obtainable if natural oxidization of tailings is allowed to occur, would appear to be fairly satisfactory."

Amalgamation of Raw Tailings in the Washoe District.—There are a number of establishments in this district engaged entirely in working over the tailings of the crushing mills. Some of the smaller ones are situated in the cañons or in the immediate vicinity of the mills which furnish their supply, but the most important are placed near the large reservoirs just described, and draw from them the material for their work. The largest of all the mills thus engaged was that of Mr. Birdsall, at Dayton. There were 35 pans in this mill, with a capacity for working from 250 to 300 tons of tailings per day.

The mill of Messrs. Janin and Baldwin has five McCone pans, with a capacity of about 50 tons per day. It is driven by steam, an engine of 12-in. cylinder doing the required work. Each pan works a charge of 4,000 or 5,000 lbs., and four or five charges per day, making a full duty of 10 tons per pan for each twenty-four hours.

Sulphate of copper and salt are supplied to the pans with each charge; the former reagent in quantities varying from 3 to 6 lbs. per ton of tailings, and the latter largely in excess, from 20 to 30 lbs. per ton. The pans are covered and supplied with steam, keeping up a high temperature. The yield obtained is thought to be about 60 per cent. of the assay value, which is said to average £3 to £3 10s. per ton. There was extracted from the tailings on an average £2 per ton. The expenses of working the tailings was £1 6s. per ton. The current cost of operation appears to be about as follows:—

	£	s.	d.
For labour	0	6	0
,, quicksilver, lost	0	4	0
,, salt	0	3	0
,, sulphate of copper . . .	0	2	6
,, fuel	0	5	0
,, castings	0	0	6
Total per ton	£1	1	0

In Avery's tailing mills, in Washoe Valley, where wood is £1 4s. per cord, the cost per ton is said to be 14s.

CHAPTER VI.

THE WASHOE PROCESS AS APPLIED IN OTHER MINING DISTRICTS.

MILLING in Owyhee District, Idaho, as practised by the Author—Passing the Ore through the Rock-breakers—Crushing in the Battery—Settling of the Pulp in the Vats—Amalgamation in the Pans—Working the Settlers, Agitators, and Concentrators—Gathering the Amalgam and Retorting it—Saving of Slimes and subsequent Treatment—Milling in White Pine, Nevada—Milling at the Montana Company's Mines—At Silver Reef, Utah—At Pioche, Nevada.

Milling in Owyhee District, Idaho.—Having had the charge of the works at the Golden Chariot Silver Mine in this district, I am enabled to give an account of my own experience there in applying the Washoe process.

The character of the ore won from the mines located on the crest of War Eagle Mountain—a granite range intersected by several volcanic dykes—resembles those of the Comstock only in regard to the proportion of gold they carry; otherwise they are not of so complex a character. The silver I found to occur generally as a sulphide with some chloride of silver, and on the whole the returns were satisfactory. On an average I obtained 90 per cent. of the assay value of the ore.

The mill was composed of a 20-stamp battery with stamps weighing 650 lbs., sixteen amalgamating pans, eight settlers, four agitators, and four Hungerford concentrators. The reduction was conducted as follows:—

1. *Passing of the Ore through the Rock-breakers.*—The ore on being brought from the mine was delivered by teams in front of the rock-breaker. I always set the jaws of the

breaker very close, so as to break the rock as fine as possible, and it made as much as 5 tons per day difference in the increase of the reducing capacity of the mill by having the ore ground finely in the rock-breaker before delivery to the stamping battery. As a rule the ore should be as uniform in size as possible, and should be fed into the mortar as regularly as possible. "The ore should be so fine that a single blow of the stamp will be sufficient to shatter thoroughly each piece of ore," and to get rid of it as soon as crushed.

The stamps had a drop of 8½ in., and ran at a speed of 90 drops per minute, and using a coarse screen, 45 tons of ore were crushed in twenty-four hours.

2. *Crushing the Ore in the Battery.*—The feeding was done by hand, and although there are many excellent self-feeders in use, I have always been partial to hand-feeding, especially in mills where intelligent and experienced men can be had for this work. Money, in fact, should be no object in paying a good feeder; he will earn for the company his daily wages twenty times over if he does his work properly. Of course an automatic feed is more economical than to have the feeding done by a man who is negligent, lazy, or inexperienced. For doing this work, heavy, stout men should be rejected; they cannot stand the jarring in front of the battery, and it even affects the nerves of a slight wiry-built fellow to be constantly shaken during twelve hours in front of a battery. The feeding should be done by three eight-hour shifts, one man feeding twenty stamps. If a small mill, say of ten stamps or less, is to be supplied with ore, self-feeding is more economical than feeding by hand as performed by ordinary workmen; but if the mill is pressed with work and the pans are of sufficient capacity to crowd the battery, the self-feeding apparatus is not so good as a skilful feeder.

An accustomed feeder shovels in the ore according to the sound of the stems; his ear gets accustomed to the ring of metal against metal, or metal against ore, and he will always feed *low*. He must take care not to allow the stems to fall on the

naked die, as it will cause the stems to break off. Such breakages can be easily repaired by cutting off above the break and welding on a piece of a bar of rolled iron, which is subsequently turned off in a lathe. The feeder's duty is to keep the shoes off the die, and therefore to have always a certain amount of ore on the dies, so that the stamp shall not fall on naked iron. An experienced feeder will hold the stamp in the hand while the same drops, and tell if the same strikes "low" or "high."

It is difficult to say exactly if heavier stamps and less drop would have given better results on Chariot ores; but with the speed of 90 it was necessary to keep the battery in good trim, have all bolts tight, nuts secure, guides snug, as otherwise the battery would shake apart and breakages would result.

The supply of water to the battery is increased if the ore is of a clayey nature, and there should be no more water than is necessary to keep the screens clear; and the launders leading from the battery to the tanks ought to have sufficient fall so as not to choke up. With clay ores one must be careful not to choke up the mortar. Again, by using too much water a large amount of slimes is carried out of the mill, and it is advisable with such ores to use a No. 4 coarse-punched Russia iron screen.

No battery amalgamation was required for these ores. The loss of float gold was exceptionally small, as was shown from the assays of the slimes, which carried only a very small percentage of the gold contained in the ore, and had any float gold been carried away some of it certainly would have settled in the slime pits. Where the Washoe process is used and the pulp is treated in pans, the amalgamation in the battery is not required.

I subsequently introduced the practice of putting quicksilver into the battery, especially when running on ores rich in gold, as occasionally rich streaks would be encountered which would show coarse gold in abundance. Although I tried all varieties of screens—punched with round holes, slots (either horizontal or vertical), brass wire, &c.—the result seemed always the same.

I consider that a double discharge of the mortars would have been an advantage. I generally employed 40-mesh brass wire screens. From the mortar the ore was discharged into a wooden box, and flowed through a trough which carried it to the settling vats.

The water was supplied to the batteries by a 2-inch ordinary gas pipe, which passed along the line of batteries just above the feed slots, and over each mortar five small faucets were inserted corresponding to the five stamps, so that a small stream was discharged from each faucet. Probably one large faucet would have answered the purpose quite as well. In winter—and the winters are severe and long in that region—the battery water was heated to a temperature of about 80° F. by a heater specially constructed for that purpose.

As the shoes and dies were made in a small local foundry, and were not of the best iron, they wore out very rapidly and had to be frequently renewed, occasioning stoppages; but breakages of stems, cams, or tappets were of rare occurrence, owing to the good and careful attention of the workmen. I should judge that the wear of the shoes amounted to over 1½ lb. per ton of ore treated. In some cases steel shoes are extensively used.

I consider that the wear of iron from shoes and dies is amply repaid in the subsequent amalgamation in the pans, as the resulting finely-divided iron, disseminated through the ore mass, plays an important rôle in the Washoe amalgamation, perhaps more so than is usually attributed to it. I have never seen the subject mentioned before, but my own opinion is that the finely divided iron in the pulp is a very important reagent during the reaction which takes place in the pan.

3. *Settling of the Pulp in the Vats.*—In front of the battery were a series of vats, and the crushed material as it discharged from the battery was carried by means of a launder into one of these vats; and when this was full communication with the battery was cut off and the flow directed into another tank. The waste water was allowed to circulate through all the empty

tanks, so as to settle as many as possible of the mineral particles which it held in suspension, and on running out of the building was discharged into the slum pit, located near the mill.

I may point out here that the system should be so arranged that as each tank is emptied of sand, the escape or waste water can be turned into it. Each tank thus becomes in turn the final one of the series and receives all the water after settling through all the other tanks. There should never be more than three tanks full of sand; the remainder should be used for the settling of the slimes in the water.

In some mills the settling capacity of the vats is large enough to allow not only the sands but also the slimes to settle, and the pulp is treated together with the slimes. Generally before charging the pulp into the pans, the same is shovelled into small heaps on the platform in front of the pans: the platform slightly inclines towards the vats, so that the surplus water can drain back into them.

4. *Amalgamation of the Pulp in the Pans.*—From the tanks the pulp was shovelled into cars running on light rails and dumped into the pans, where first of all the pulp was ground fine by lowering the muller.

A proper consistency of the mass is important, and also the degree of heat. The grinding brightens the gold, and prepares the silver ores for the chemical reaction by causing their perfect trituration. The process may be thus described:—

After charging the pan the muller is lowered, hot water added, and the steam turned on. Direct steam into the pan is better than heating by jacket or double bottom. The charge must be heated nearly to boiling point. At the commencement of the grinding operations it is best to have the pulp thin, and after a couple of hours' grinding the pulp will acquire the proper consistency for receiving the quicksilver.

The quicksilver will, after its speedy division into small globules, occasioned by the grinding and the heat, be diffused through the whole mass. A sample of the pulp taken out on a

thin wooden spatula should show particles of uniformly disseminated quicksilver. Some of the globules will be microscopic; but from an ounce of the pulp, washed in a horn spoon, a good-sized globule of quicksilver should be collected. The pulp should be thick enough to carry the globules of quicksilver in suspension.

Ten pounds of salt is added as soon as the pan is charged, and two pounds of sulphate of copper half an hour after. As soon as the mass has been heated to 180° Fahr. steam is shut off. It is better to lower the muller gradually, and not at once, for the grinding operation. Should the pulp get too cold a little more steam is allowed to escape into it.

After two hours' grinding the quicksilver is added, say 200 lbs. to 1,200 lbs. of ore. The grinding operation is now continued for one hour more, then the muller is raised and the pan run for three more hours. Fifteen minutes before drawing the charge sufficient water is added to thin the pulp thoroughly. This prepares the charge to flow readily out of the pan and also stirs up any pulp that may be moving sluggishly. At the same time the mass is considerably cooled.

I invariably found that an addition of salt and sulphate of copper increased the yield of silver. The average yield from the pan amalgamation was 80 per cent. and over, and as the slimes would at the end of the month increase this production 8 to 10 per cent., I can claim that these ores were very successfully treated, yielding 90 per cent. of their value. They are very docile ores, and the gold was comparatively coarse, which accounts largely for this success.

I also found that grinding for three hours instead of four gave equally good results, reducing the wear of iron and the power consumed. The most important point, I found, was to keep the quicksilver always bright, clean, active, and in good order. In working an ore that fouls the quicksilver it is not practicable to keep the quicksilver clean in the pan; it should be at least put in perfect order before it is again used for another charge. For cleaning quicksilver, sodium amalgam, caustic potash, dilute acids, cyanide of potassium are used.

The pans were ordinary flat-bottomed pans of the Wheeler pattern; each pan was charged with from 1,200 to 1,300 lbs. of ore; and the work arranged so as to charge and discharge every six hours, making the capacity per pan something over 2½ tons. It is best to have the pans in front of the settling tank and at a convenient level, so that the pulp cars can dump the ore right into them.

I found that in grinding the ore it was best to add the quicksilver shortly before the grinding was finished. After two (or two and a half) hours the muller should be raised and the quicksilver added; this avoids the flouring of quicksilver. There might be with certain ores economy in crushing finer in the battery by using 50 or 60 mesh screens, and thereby delivering the ore in a better condition for amalgamation. It is obvious, however, that such a system will to a certain extent decrease the crushing capacity of the battery.

The charge should fill the pan about half full, and the pulp ought not to be too thick or too thin, but of a consistency to hold the quicksilver globules in suspension. I gave the muller about sixty revolutions per minute. The water added to the pan is previously heated by steam, but I must caution mill-men against discharging the exhaust steam into any water used about the mill.

With a proper consistency of the pulp and an even distribution of the mercury globules, the current created in the pan will carry them through the central opening, in and under the muller, out and around the periphery up to the surface again, and a perfect contact of metallic particles and quicksilver will be insured.

5. *Working the Settlers, Agitators, and Concentrators.*—When the pans are discharged a stream of water should be directed into the pans, so as to rinse them out well. Everything then flows into the settlers, where, through the dilution of the pulp, the quicksilver settles to the bottom, whence it flows out, free from sand, through a siphon, into a kettle outside. When the charge is drawn it should not fill the settler,

but a spray of water falling like a rain shower is turned on, and when the settler is full the water is turned off, and the stirring arms are revolved for an hour. This allows the floured quicksilver to collect and settle. Then turn on plenty of cold water, and let the settler discharge through the top plug hole as long as possible. The operation should be so timed as to reach the bottom hole of each individual settler only just in time to receive the next charge. The settler will never choke with heavy sand if the pan has ground well and the driving belt is in good shape. In the settler accumulate some coarse sands, some sulphurets, amalgam, quicksilver, and iron from the pans; and once a week or oftener the settler should be cleaned out and the concentrations re-worked in the pans.

I generally discharged two pans at once into one settler, and while charging these two pans with ore again, I emptied the two following ones. While working in this manner no part of the machinery was idle, and it kept the working men steadily employed without overworking them.

In some mills all the pans are discharged at once, which involves a rush of the workmen to charge them again. It also has a tendency to cause irregular working of the engine, as all at once the full power is required to run the pans, and then suddenly, when the pans are discharged simultaneously, less power is required—which means that the fireman is obliged for an hour or so to maintain his full pressure while there is no consumption of steam.

The settlers discharge into agitators—generally two settlers into one agitator—and a constant stream of water should run into them. Here there will be found some coarse sand containing a little quicksilver, amalgam, sulphurets, and a quantity of iron; but the saving is very small. The floors throughout the mill should be kept clean, and the whole mill as neat and free from dirt as possible. No loose quicksilver should be found in the floors, on the tables, or anywhere. All drains should lead into the agitators; and the quicksilver floor, unless the weather be too cold, should be washed with a hose every day.

I found it did not pay to concentrate the tailings, although

they were passed through the Hungerford concentrators. The ore was ground finely in the pans, so that if any metal escaped in the tailings it did not settle. The diluted pulp from the agitators was siphoned by means of a 2-in. pipe into the concentrators. Hungerford concentrators I found very good for slimes and slimy ores, since the shaking collects the floured and slime-coated quicksilver very well. After leaving the concentrators the tailings were run over blanket sluices 250 ft. long, and they were washed once a week, but only little was saved on them. The floured quicksilver collected on the concentrators contained also some gold and silver.

6. *Gathering the Amalgam and Retorting it.*—I have described in my previous account of the Washoe process how the amalgam is collected in pointed canvas bags, and I gave a description of the retorts.

The process of retorting is very expensive, it being necessary to heat the retorts during a certain length of time to a bright cherry-red heat; otherwise it is difficult to drive out the last traces of quicksilver, and this cannot be done unless some of the bullion melts. The cast-iron retort, being exposed repeatedly to such heating, becomes spongy and rotten, and the bottom bulges out and the whole retort loses its shape. This necessitates the retort being turned round, so that the bottom shall be on top, and occasionally the retort bursts and the quicksilver vapours go up the chimney and are lost. Such a volatilization of quicksilver may cause a frequent loss of £20 to £40, and even with the greatest care this is apt to happen several times during the year. The only remedy is to brace the retort well, not to fire above a certain heat, and to leave 1 or 2 per cent. of quicksilver in the bullion. This, naturally, is lost in the subsequent melting, but much inconvenience is avoided in resetting retorts or paying for new ones.

I always let the retort stand closed for twenty-four hours before taking out the retorted bullion, which should not be hot, as the uncondensed quicksilver fumes are dangerous. It was my custom to commence firing under the retort at six P.M.,

when the night shift took their turn, and by six A.M. the work was done. The lid was left on luted till the next morning; this gave a chance for the contents to become perfectly cool and all the quicksilver to condense.

The bullion was then withdrawn in my presence from the retort, weighed on platform scales, and locked up till ready for the melting pot. I generally got 200 lbs. of bullion from 1,300 lbs. of amalgam. It would be of a spongy nature unless the last firing melted some of it. One steady run of 5,123 tons of rack gave $\$348,052^{75}$ or about £70,000, giving a yield of $\$67^{93}$ per ton, the proportion of gold in the bullion being nearly one half in value.

7. *The Saving of Slimes and subsequent Treatment.*—The tailings from our mill were not saved, as they were not rich enough to pay for subsequent treatment; they assayed from 16 to 24 shillings per ton. In most mills they do save the tailings, and after an exposure to atmospheric influences for several years, the mineral particles become decomposed or oxidised, and from 50 to 60 per cent. of their silver can then be recovered. We saved only the slimes, or those fine ore particles which were carried away by the battery water and were deposited in the slime pit outside the mill. The slimes (which are known also in this locality as "slums") usually contain less precious metal than the ore, and especially their gold contents are very much lower.

The percentage of slimes collected varied with the amount of clay, the quantity of water used, and the method of settling; but they generally amounted to from 2 to 3 per cent. of the amount of ore crushed.

They were brought into the mill on a tramway, worked by a rope-and-bull wheel, and I generally added a certain quantity of them to the ore in the pans, but did not make it a practice to work them separately.

Milling in White Pine, Nevada.—The silver-bearing mineral in this district is chiefly chloride of silver, and conse-

quently very easy to work, yielding a very high percentage of the silver contents and very pure bullion.

As chloride of silver is a very light substance, which easily floats away on water, several of the mills in the district, when first set up, were arranged for dry crushing, and although the quantity of ore crushed was smaller than by the wet process, the increased yield of bullion compensated for the loss of rapid crushing by water. The lower-grade ores were worked wet, as the loss is proportionately smaller, and when the grade of the ores in the whole district diminished, after the rich surface deposits were worked out, all the dry-crushing mills were turned into wet-crushing mills.

Pan amalgamation was conducted in the same manner as mentioned in describing the Washoe process. Why salt should have been used, even in considerable proportions, with ores which were a chloride, I never could clearly understand; but it was a force of habit among mill-men to use "copperas and salt" in those days, and so it was added, whether beneficial or not.

The milling results averaged from 70 to 80 per cent. of the assay value of the ore, and in some of the dry-crushing mills 90 per cent. was obtained. There was no gold in the ores, and the bullion produced averaged 950 fine; some was nearly pure silver, about 997 fine.

The average cost of milling in the early days of the district was £2 per ton, but the cost of labour and transportation was very high. The average assay value of some 30,000 tons of ore crushed in 1869 was £16 per ton, and taking the average returns at 85 per cent., the returns in bullion amounted to about £13 10s. per ton.

There is a great similarity in the treatment of the silver ores in the different districts where the wet process is employed, although occasionally some slight alterations are encountered.

Milling at the Montana Company's Mines.—The Montana Company's mills are located at Marysville, Montana, and

consist of one 60-stamp mill, one 50-stamp mill (shown in plan, elevation and section, in Figs. 43, 44, and 45), and one 10-stamp mill. The capacity of the mills is 70,000 tons per annum, and as a whole they may be considered as amongst the best structures of the kind in the United States.

The milling capacity per annum of the 50-stamp mill is 30,000 tons, and the preliminary operation of crushing in breakers and batteries is the same as described in the previous chapters. In this mill the pulp undergoes the following treatment:—

1. The pulp from the front of the mortar boxes flows over amalgamated plates for the purpose of depriving it of the free gold it may contain.

2. The pulp from the amalgamated plates passes over twenty Frue vanners, which collect various ores more or less rich in the precious metals.

3. The pulp, deprived mostly of free gold and concentrates, now passes to the amalgamating pans, where it is worked hot in charges of about 2,100 lbs. each, with the addition of a given weight of quicksilver, salt, sulphate of copper, and sulphuric acid. In this operation most of the silver, with a limited quantity of gold, becomes amalgamated with mercury.

4. From the pans the charge goes to settlers in which the amalgam is collected. From the settlers it is discharged into agitators in which a small quantity of coarse sand, carrying mostly gold, is retained, together with some particles of amalgam which had escaped the settlers.

5. From the agitators the pulp, largely impoverished of its metallic contents, passes a second time over twenty Frue vanners, which operation secures a small additional quantity of inferior concentrates.

6. The pulp from the end of the vanners, or the tailings, is now deposited in dams to undergo an oxidation process, with a view to further treatment hereafter.*

* The Frue vanners in front of the copper aprons are not shown in the plan of the Montana Company's 50-stamp mill, as they were placed in position subsequent to the production of the plans.

FIFTY-STAMP GOLD AND SILVER MILL
Erected for the Montana Company, Limited,
Marysville, Montana, U.S.A.

SCALE— 0 10 20 30 40 50 60 Feet.

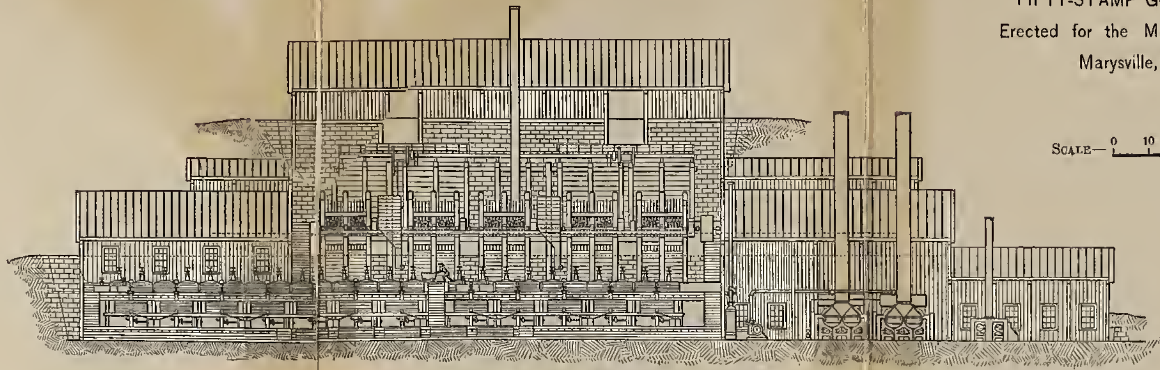


Fig. 43.—Elevation.

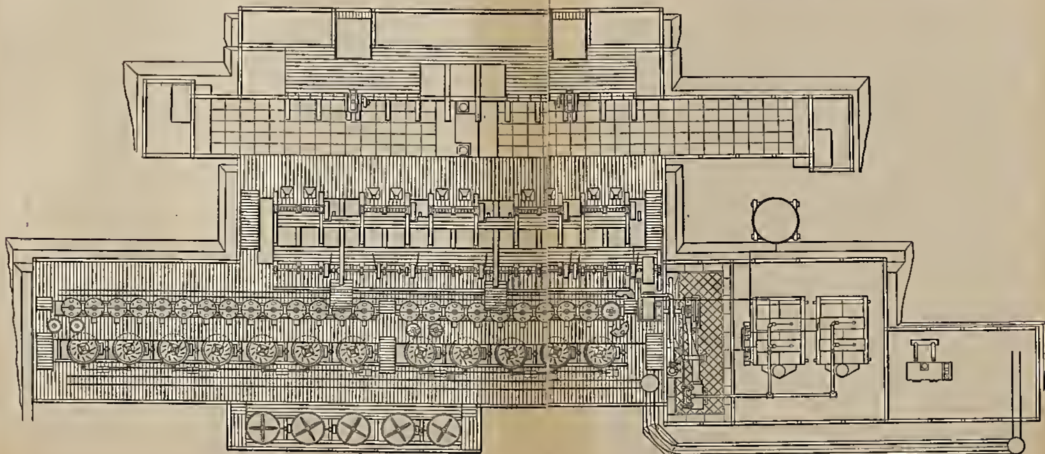


Fig. 44.—Plan.

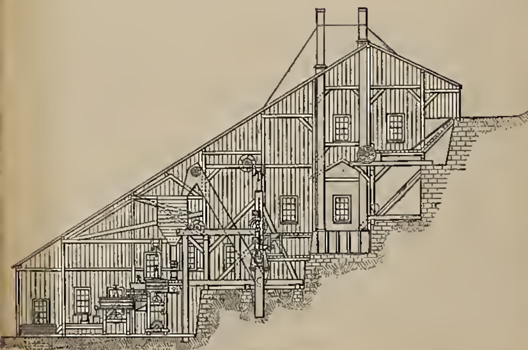


Fig. 45.—Section.

p. 126.

In the 10-stamp mill the process differs only in one respect—the pulp is twice concentrated on Frue vanners before it is charged to the pans, and not after it has left the pans.

Special care and vigilance are exercised in testing the results of the milling operations by taking samples for assay. Samples of the pulp are drawn from the lip of each battery box once every hour, and these hourly samples are thrown into a common receptacle, and at the end of twenty-four hours resampled for assay. A sample from the lower end of each amalgamating table is taken every hour, and twenty-four samples disposed of in like manner. A sample from the end of each of the twenty-four vanners in the 50 and 10 stamp mills is taken every hour, and the aggregate number (576 samples) resampled for assay at the end of twenty-four hours. A check vanner sample is also taken from the whole of the pulp flowing from the ends of the vanners through a common launder every hour, and resampled and assayed as described. Three settler samples are taken from every charge of about 2,100 lbs. amalgamated in the pans; these samples are collected together, resampled, and assayed at the end of twenty-four hours. In the tailings vanner house, containing twenty-four vanners, a sample is taken every hour from the end of each machine, then resampled every day and assayed. Samples from the tailing dam are also taken from time to time. The various samples are assayed daily for bullion, and subsequently the gold is parted from the silver in the ordinary way. The gold is valued at a standard price of $20·67, and the silver at $1·2929 per ounce troy. The decline in the value of silver from this standard sum has now reached 91 cents, or a difference of, say, 1s. 7d. per ounce.

At these mills, the concentration of the battery pulp before amalgamation has been attended with most favourable results, namely, the sulphides (obtained as concentrates) render the pulp fairly clean for the pans; the bullion, instead of being low, is brought to a high degree of fineness; and lastly, it has led to a very important and valuable saving of

mercury in the pans, and in melting the crude bullion to a material reduction in the weight of granulations, or metallic slag.

The new 60-stamp mill was designed for the treatment of low-grade ore. It includes three stone-breakers, sixty heads of stamps, amalgamating plates and tables for obtaining free gold, and twenty Frue vanners for collecting concentrates. The disposition of the mill is such as to admit of the introduction of pans and settlers, should these appliances at any time be required.

Milling of Pure Chloride Ores in Silver Reef, Utah.—A very extraordinary occurrence of silver ore—which may, indeed, be characterised as a geological phenomenon—came under my observation in Washington County, Utah, where silver mines occur in a sandstone ridge, running parallel with and at a short distance from the main Wahsatch Range for about one hundred miles.

The Harrisburg district is the principal mining centre, three hundred miles south of Salt Lake City, in the Rio Colorado basin. The ore is a sandstone, of a chocolate colour, not differing materially in appearance from that of the country rock, which, according to all indications, belongs to a very recent geological formation. The ore contains disseminated in its pores fine chloride of silver; but where organic materials are embedded in the ore, such as leaves of trees, stems, and even animal remains—the silver is present in a pure metallic state, and some choice specimens have come from those mines, like leaves of trees incrustated with silver, which no work of man could imitate or reproduce.

Geologists differ as to the origin of silver in this remarkable district, but I venture to think that the sublimation theory cannot be upheld in a case like this, where there is so much evidence that the silver was precipitated from solutions.

The ore is remarkably uniform, both as to quantity and quality, and since 1877, when milling operations were com-

menced, silver to the value of probably £2,000,000 sterling has been produced in the locality.

In the Stormont Mine the ore is found within a zone from 10 to 100 ft. thick, often in association with fossil remains and petrifactions of reeds and rushes, and is bounded by red sandstone above and white sandstone below. The deposits vary in length from 50 to 100 ft., are from 100 ft. to 300 ft. in depth, and from a few inches to several feet in thickness; sometimes they are connected with each other by feeders or stringers. The ore averages about $30 or £6 per ton.

The silver-bearing part of Silver Reef is known to be fifteen miles long, but is to a great extent unexplored, and in all probability there are many deposits there which when opened up will be profitably worked.

The sedimentary sandstone impregnated with silver chloride crushes so easily that a 750-lb. stamp will crush through a 40-mesh screen an average of from 7 to 8 tons per twenty-four hours. The mills in the Silver Reef district are all small, containing only five or ten stamps; and although they contain as much as twelve pans, with a capacity of 1½ tons per charge, the battery capacity is in all cases greater than that of the pans. Some of the mills have averaged the month through 8½ tons to the head of stamps, nor does that appear to be a maximum limit. The ore under the stamp disintegrates and passes rapidly through the screens as fine sand; indeed, it seems probable that Cornish rolls would crush this ore fine enough for amalgamation, and would do so with wonderful rapidity. As the ore is also remarkably pure, no impurity except a little copper, which occurs in a few of the mines, is found in the bullion, which averages from 0·950 to 0·980 fine. The cost for chemicals is also extremely low, and but very little bluestone is used. Considering the fact that the mills are so small, and that some of the items are therefore necessarily high, the cost of milling is the lowest of any silver ores in the country.

The tailings vary greatly in richness according to the character of the ore milled. From sandstone ore they will carry $3 per ton, while from the shale ore they may run $10 or more.

The subjoined table (for which I am indebted to Mr. R. P. Rothwell's Report on the Cost of Milling Ores in Utah) gives details of cost for a year's working at three of the mills :—

Per ton of 2,000 lbs.	Christy M. & Mg. Co. 14,249 tons.		Stormont Co. 9,983 tons.		Leeds Co. In 1878. 12,064 tons.	Leeds Co. 4 months in 1879. 4,679 tons.
Labour and Salaries		$2·85		$2·97	$2·20	
Bluestone	2·1 lbs.	·31	$1\frac{3}{4}$ lbs.	·26	3·22	
Mercury	1·22	·58	1·13	·57		
Salt	25·8	·51	20·	·29½		
Fuel		1·31	*	·45½		
General supplies		·87		·45		
Incidentals		·41		·12		
		$6·84		$5·12	$5·42	$4·12
Hauling ore to mills		·73		2·00	·32	·25

Milling Base Ores at Pioche, Nevada.—One of the most productive silver mining-districts in Nevada—namely, Pioche, in Lincoln County—gave to metallurgists at the beginning a striking illustration of the difficulties so frequently met with in finding the proper method for dealing with newly discovered mines and special characters of ores.

In the belief that the ores were of the smelting order, large and expensive smelting works were erected. The attempts at smelting, however, proved abortive, and were speedily abandoned ; the cupelling of the few tons of lead bullion extracted was also a failure, and the bullion was shipped in a semi-refined state to San Francisco. Similar attempts at smelting, made in furnaces of a very primitive construction, by individual miners, proved equally unsuccessful. The discovery was then made that the ores did not contain enough lead for smelting purposes, and that other means would have to be found for their reduction.

Several lots of the ore were shipped to White Pine and

* Stormont mill is driven by water power.

to San Francisco, with a view to testing its adaptability to the amalgamation process. The ore shipped to Hamilton, White Pine, yielded, by the ordinary mill process, over $500, or £100, per ton. The ore was "free," and gave bullion over 950-thousandths fine. A second lot yielded only $190 per ton, although assaying very high; this was of the "base" description. Attempts made at amalgamation in San Francisco on ores of the normal (base) character proved also unsuccessful, yielding only from 45 to 55 per cent., according to their richness.

The ores of the district, it was found, varied in character. While the general and normal quality of ore contained enough lead to be called base, and to prove refractory in amalgamation with quicksilver alone, there could be found in certain spots on the various ledges chimneys of very free ore containing little or no lead, and working readily without chemicals from 75 to 80 per cent. From any point of the ledges, pieces of almost pure carbonate of lead and galena could be extracted. The average percentage of lead in the ores as they came to the mill did not fall short of 3 to 5 per cent. The "free" ore mentioned above contained 75 to 80 per cent. of the silver as chloride; the baser quality contained only from 40 to 45 per cent. of chloride. A series of chlorination tests showed that the percentage extracted by amalgamation, with quicksilver alone, corresponded invariably with the amount of chloride of silver in the ore; so that amalgamation in this case extracted only the silver chloride and nothing else.

Subsequent experiments made on the base ores by the Washoe process succeeded beyond expectation, and they readily yielded their silver on the addition of salt and bluestone.

A 30-stamp Washoe mill was then erected at Pioche, with stamps of 650 to 750 lbs., eighty drops per minute; fourteen pans, calculated to hold 2,800 lbs. of pulp each seven settlers; and four agitators; two Hungerford concentrators and two revolving buddle concentrators, each 20 ft. in diameter. The ore was worked up to over 82 per cent., the ores assaying $130 per ton. The average fineness of the bullion was below 700,

containing, of course, lead and some copper, after passing the amalgam through the following process.

A simple method of extracting the greater part of the lead from the amalgam, and consequently from the bullion, was employed. The quicksilver and amalgam, after leaving the settlers, was strained in sacks suspended in a large box filled with water, heated with steam by means of a half-inch pipe. Lead amalgam, at the temperature of boiling water, remains liquid, and will therefore strain through with the excess of quicksilver; a certain amount of silver and of copper amalgam also passes through. This was now run off into a smaller box, cooled with water, and when cold strained in the usual way, leaving an amalgam of lead containing a small amount of the other metals. This lead amalgam when retorted gave bullion containing from 6 to 20 per cent. silver, very little copper, and only a trace of gold. The amalgam remaining in the first sacks gave bullion from 550 to 680 fine in silver, and finer in inverse proportion to the amount of copper in the ore.

From a charge of ore of normal character the bullion extracted by different methods of working would be nearly as follows :—

Amalgamated	without chemicals . . .		300 to 350 fine
„	with salt and bluestone, and not strained in hot water . .		400 to 450 fine
„	with salt and bluestone, and strained in hot water . . .	1st amalgam.	550 to 680
		2nd „	60 „ 200

The ores contain on an average $5 in gold to every $100 in silver. This proportion is very constant. Of this gold, from 45 to 55 per cent. is extracted. The bullion contains from 0·0003 to 0·0015 parts gold. Occasionally a bar will contain as high as 0·003 parts, and at other times so little that it is not taken account of. The $130 ore contained from 40 to 50 per cent. of the silver as chloride, the remainder was a sulphide and probably some oxide.*

* Mr. R. W. Raymond's Reports on Mineral Resources of the United States.

The ore was admirably adapted to concentration. The "pay" appears to be mainly in a "heading" of gray carbonate of lead, very easily separated in the sands. The gangue is quartz, the country rock quartzite. All the tailings, after leaving the agitators, flow into a tank from which they are raised by a china pump to a second tank, and from there distributed over two concentrators. The tailings assay on the average about $25, and the concentrations will range from $150 to $300, according to the amount of water used in concentration.

The average loss of quicksilver was 2½ lbs. with bullion 800 fine. With bullion 500 to 600 fine (after passing through the hot-water straining process) the loss of quicksilver amounted to 4¾ lbs. per ton. This was due probably to the formation of chloride of lead and subsequent formation of subchloride of mercury. Another source of loss was the formation of floured lead amalgam, which contained little quicksilver, having the dull appearance of lead, and would float off in flakes on the top of the water. Lead unites with quicksilver in greater proportion than either silver or copper.

There was a rapid destruction of muller plates and castings in the pans, which were strongly attacked by the chloride of copper.

The proportion of retorted bullion to amalgam on the Comstock, White Pine, and Idaho ores, is as 1 to 5½—6; in amalgam containing a large amount of copper as 1 to 7—7½; and in very base lead amalgam as 1 to 4.

From the foregoing will be seen the importance of maintaining the proper proportions of salt and bluestone on all ores which are worked by the Washoe process.

The Boss System of Amalgamation.—It has been explained in previous chapters that all the pulp, as it is discharged from the battery, is collected in vats and then charged by manual labour into the pans. The idea that this disagreeable and costly handling of the pulp might be avoided is very old, but Mr. M. P. Boss, an American metallurgist, was the first to

put it into practice, and I am informed that his plan has succeeded with certain classes of ores.

He conveys the pulp directly from the battery to the first of the pans, which, as well as the settlers, are all on the same level. In ordinary mills the settlers are on a lower level and the pulp discharges into them by gravitation. But by the Boss system, the pans and settlers being all on the same level, they have to be so connected by means of syphons that the pulp may flow freely from the one to the other. In the first pan the pulp is submitted to the grinding operation, and is then drawn into the second pan, where the grinding is continued. It is then drawn—again by a syphon—into the third pan, where sulphate of copper and salt are added, and a portion of the quicksilver. From there it is drawn into the fourth pan and the balance of the quicksilver added; in the fifth pan some lime is added for the purpose of cleaning the quicksilver; and from the last pan the pulp is drawn into settlers, where the amalgam is collected.

This system requires working with a very thin pulp, and the battery is fed with hot water. The pans are heated by false bottoms, and are so arranged that any of them may be thrown out of gear when repairs or stoppages are required. In that case, syphon connection is established between the other pans, so that no interruption of the work takes place.

With chloride ores the Boss system appears to have given satisfactory results, but I have no definite information as to its working with sulphides, which are not so tractable as chloride ores; nor have I been able as yet to get reliable data as to the development of the Boss system on complex ores. I consider its practical application a matter of importance in silver milling, as if successful it would allow of the automatic working of the ores, dispensing with manual labour.*

* A description of the plant employed in carrying out the Boss system will be found in Chapter XV., *post*, pp. 331—342.

CHAPTER VII.

CHEMISTRY OF THE WASHOE PROCESS.

Composition of the Comstock Ores—Mr. Hague's Experiments on Action of Mercury and other Reagents—Chemical Action of Sulphate of Copper and Salt—Iron as a Reducing Agent—Action of Quicksilver—Practical Conclusions.

Composition of the Comstock Ores.—The ore of the Comstock is composed of quartz, as the gangue, while the metal-bearing minerals of common occurrence are blende, galena, argentite, silver, gold, iron pyrites, copper pyrites; and minerals of rare occurrence: stephanite, polybasite.

Chemical Action of Mercury and other Reagents.—From experiments which were undertaken by Mr. Arnold Hague—with a view to ascertain, as far as possible, the action of mercury upon the minerals of the Comstock ores, as well as the action of such chemical agents as are employed in the amalgamation process, or may be formed during the operation in the pan—the following results were ascertained:—

Mercury and native silver, when rubbed together, unite easily.

Mercury and chloride of silver, the latter prepared in the wet way, when brought in contact form amalgam and chloride of mercury.

Mercury and argentite. The mineral was first pulverised and mixed with a little fine sand, the metal added, and the mass allowed to stand for some time, occasionally rubbed together in a mortar. Amalgamation ensued; it was, however, imperfect, much of the mineral being unacted upon.

Mercury with stephanite and polybasite, under the same conditions as the last experiment, gave similar results; the decomposition, however, appeared to be more complete, probably owing to the more finely divided state of these minerals than the more sectile argentite.

The above experiments with native silver, chloride of silver, argentite, and polybasite, were repeated with mercury containing a small quantity of copper amalgam in solution. In the case of the two former there was the same action as when the pure metal was used; with the two latter the decomposition was more perfect and satisfactory.

Chloride of silver, argentite, and stephanite were each subjected to the action of mercury and fine metallic iron, with a constant application of heat. The energy displayed by the mercury was much more marked than when employed separately. In the case of the chloride, the decomposition was quite rapid, and the surface of the metal remained bright and clean.

Chloride of copper and pulverised argentite were allowed to stand together for ten days in the cold, with an occasional application of heat, at the end of which time a small quantity of chloride of silver was formed. A trace of sulphuric acid was found in the filtrate.

Two grammes of the pulverised mineral were also treated with a moderately concentrated solution of chloride of copper placed in a bottle, with a tightly-fitting stopper to prevent access of air. It was exposed for twenty-four hours to a temperature of 90° centigrade. Sulphuric acid and subchloride of copper were found in the solution. Chloride of silver was precipitated. After removing the soluble salts by washing, the chloride of silver was dissolved out by digesting it with ammonia. The residue gave, by assay, ·099 grammes of silver. Two grammes of the mineral produced ·1705 of pure metal; showing that, under the most favourable conditions, but little over one-half of the silver was chloridised. The application of heat greatly facilitated the decomposition.

Polybasite, after being subjected to the chloride of copper

solution at the ordinary temperature of air, also yielded a small quantity of the chloridised silver.

Argentite was exposed to the same treatment with subchloride of copper as in the last experiments. In the cold, decomposition ensued after standing several days. The residue from two grammes of the mineral, subjected to the action of heat at 90° centigrade, without access of air, gave ·1655 of a gramme of silver, showing that only ·006 had been chloridised.

Galena in a pulverised condition was digested with a strong mixture of salt and sulphate of copper, and after standing three or four weeks at the ordinary temperature was filtered. The residue exhibited, besides the undecomposed mineral, a light green oxychloride of copper and a large quantity of sulphate of lead incrusting the galena.

Blende was also subjected to a similar treatment. The solution was found to contain a considerable quantity of oxide of zinc, and but little copper. The residual blende was coated with the same oxychloride of copper already noted in the case of the galena.

Two grammes of the powdered mineral were placed in a flask, a solution of five grammes of salt and seven of sulphate of copper added, and exposed for two days to a temperature of 90° centigrade. After remaining three days longer in the cold the amount of oxide of zinc found to have been dissolved was ·2785 of a gramme. The same experiment was repeated with the addition of one gramme of iron filings. The latter rapidly disappeared, metallic copper was precipitated, but was redissolved, probably by the chloride of copper present, the subchloride being produced. Later the iron was thrown down as a basic salt. The oxide of zinc estimated in the solution was ·3250 grammes.

Iron and copper pyrites are but slightly altered by the copper solutions. In practical operations at the mill they are found in the tailings without showing any appreciable signs of having been attacked. It will be observed that in the above experiments the argentiferous sulphurets were always more or less chloridised by the action of the copper salts.

In order to indicate more clearly the relative amount of decomposition produced by the two chlorides of copper, the results are here brought together as follows: Two grammes of argentite gave ·1705 grammes of silver. After treating two grammes of the mineral with chloride of copper, the residue gave ·099 grammes of silver; after treating two grammes of the mineral with subchloride, the residue gave ·1653 grammes of silver; showing that in the former 58·0 per cent., and in the latter 2·9 per cent. was chloridised. No sulphide of copper was detected in any of the residues examined; sulphuric acid, however, was found in the filtrate in several instances.

Chemical action of Sulphate of Copper and Salt.—The action and value of common salt and sulphate of copper in the amalgamation of argentiferous ores, by what is known as the Patio process (see *ante*, p. 20), has always been a somewhat disputed question. Numerous theories have been advanced by metallurgists of long practical experience in Mexico, to account for the reduction of the sulphide of silver by the methods adopted in that country. The two which have obtained the most prominence, and which chemists have received with most favour, differ very widely in the manner the decomposition is supposed to be accomplished.

The most plausible theory—and the one now generally adopted—is that of Sonnenschmidt. He claims that the salt and sulphate of copper react upon each other and produce sulphate of soda, which is neutral in its action, and chloride of copper. The latter salt then acts upon the argentiferous sulphide, and yields chloride of silver, subchloride of copper, and free sulphur. The subchloride reduces a second portion of the sulphide of silver, and causes the formation of an additional amount of the silver chloride and subsulphide of copper. The silver salt is then attacked by the mercury, calomel or subchloride of mercury is produced, while metallic silver is set free, which combines with a second portion of the mercury as amalgam.

The following chemical equations show the reactions:

$$CuSO_4 + 2NaCl = CuCl_2 + Na_2SO_4.$$
$$Ag_2S + 2CuCl_2 = 2AgCl + Cu_2Cl_2 + S.$$
$$AgS + Cu_2Cl_2 = 2AgCl + Cu_2S.$$
$$2AgCl + 2Hg = Ag_2Hg + Cl_2Hg.$$

Bowring, an English metallurgist, on the other hand, denies that any of the sulphide of silver is chloridised, and asserts that before amalgamation takes place metallic silver is first produced. He claims that chloride of copper, in contact with mercury, forms the subchloride of both metals. The subchloride of copper, in contact with the oxygen of the air, is converted into an oxychloride, which in turn acts upon the sulphide of silver, and liberates the metal in a free state by oxidising the combined sulphur. These reactions are expressed as follows :—

$$2CuCl + 2Hg = Cu_2Cl + Hg_2Cl.$$
$$Cu_2Cl + O = CuClCuO.$$
$$3(CuClCuO) + AgS = Ag + SO_3 + 3Cu_2Cl.$$

Although oxychloride of copper may possibly be found at times, there does not appear to be any decided evidence that such is the case in practical operations or that it decomposes the sulphide of silver, while the experiments already recorded show conclusively that both the chlorides of copper, under favourable circumstances, do chloridise the argentiferous sulphurets. The experiments, however, seem to indicate that the action of the chloride of copper was much more intense than that of the subchloride.

The action exerted by these two reagents in the pan would appear clearly to indicate that the benefits derived from their use are partly to aid in converting the sulphide into chloride of silver, as in the patio, and partly to decompose such minerals as are but slightly attacked by the mercury. In the Washoe process, however, the large quantity of iron present must tend greatly to produce subchloride of copper almost as soon as the chemical agents are thrown into the pulp.

Notwithstanding the importance of common salt and sulphate

of copper in the patio, and, under certain conditions, in the pan, their value must be considered as only secondary in the decomposition of a large proportion of the Comstock ores. The advantages derived from their use are shown to be exerted chiefly upon such minerals as blende and galena, which are but slightly attacked by the mercury. But the amounts employed are in most cases too small to effect any very favourable results. On the other hand, if a sufficiently large proportion of the reagents are consumed in the pulp, in order to produce the beneficial returns, it is always at the expense of preserving the necessary purity of the mercury.

The quantity of salt deemed necessary by mill-men varies from ¼ lb. up to 7 or 8 lbs. per ton; scarcely any two establishments have the same rule. Its action upon the ore, without sulphate of copper, in producing any marked results may well be doubted.

The consumption of the sulphate of copper also depends upon the ideas of the amalgamators, but the amount does not differ so widely as in the case of the chloride of sodium. It ranges from ¼ lb. to 3 lbs. per ton.

The addition of the sulphate without salt has also become a common practice. The opinion among those who work their ore in this way is that it gives a little better yield than when mercury alone is employed, particularly when the ore indicates the presence of galena in any considerable amount, in which case it is said to quicken the mercury and render it more energetic.

Continued experience appears to determine this fact with a considerable degree of certainty. In working ores containing only a small percentage of lead the quicksilver very soon becomes dull and inactive, or, as it is technically termed, it sickens, and the yield from the pan is consequently low. Lead is one of the most deleterious metals in destroying the amalgamating energy of mercury, and at the same time is very rapidly absorbed when the two metals are brought into contact. Sulphate of copper possesses, to a certain extent, the property of expelling lead from mercury, copper being amalgamated and sulphate of lead formed at the expense of the sulphuric acid of

the copper salt. If a concentrated solution of sulphate of copper be allowed to stand upon lead amalgam the action takes place quite rapidly, mercury containing lead acting much more energetically upon the copper solution than when perfectly pure.

This salt, however, does not appear, under any circumstances, to possess the power of completely driving out the lead.

Another advantage derived from the addition of a small quantity of the sulphate of copper is that mercury, under certain conditions, when exposed to the solution, forms a minute amount of copper amalgam, which causes the metal to act with a somewhat greater intensity in the decomposition of the silver sulphide than when perfectly pure.

Iron as a Reducing Agent.—In this respect, iron probably plays an important part in the pan process in bringing about the favourable results obtained. This may occur in three ways :—

First. It aids in a great measure the decomposition of the chloride of silver.

Secondly. It reduces the calomel formed during the operation ; the chlorine, combining with the iron, goes into solution, and the heavy metal is liberated. In this way it not only prevents a chemical loss of mercury but also serves to keep the surface of that metal bright and clean, which otherwise might be coated with a thin film of subchloride, which would greatly destroy its activity.

Thirdly. It undoubtedly assists directly in the amalgamation, where the two metals are brought into close contact with the easily reducible sulphurets. The successful and continued operations in Washoe, without the aid of any other chemical agents, sufficiently prove this statement. Experiments made in treating argentite and iron filings with mercury confirm the fact.

The consumption of iron from the batteries and pans varies very much, but experience on the Comstock shows that 10 lbs. of iron are worn away for every ton of ore treated.

Mercury and iron, under the proper conditions, undoubtedly are the principal agents in the extraction of the precious metals by the Washoe method. The results depend, however, in a

great measure, upon the mechanical treatment employed to reduce the ore to an exceedingly fine state of division, and to maintain, with the proper degree of consistency, a constant agitation of the entire mass; the essential conditions of the amalgamation being that the mercury should be thoroughly incorporated in the pulp, and every particle of the reducible minerals brought in direct contact and triturated with the metal, in the manner so well accomplished by the friction and grinding action of the pan. The mercury should also at all times retain a bright, clean surface, free from any film of metallic salts, such as subchloride of mercury or sulphate of lead, and any coating of oil or grease. The slightest tarnish appears to retard very greatly the activity of the metal. The iron seems to act as an electro-chemical agent, the immediate contact of the two metals, aided by heat and friction, causing a local electric current, which renders the amalgamating energy of the mercury much more intense.

Action of Quicksilver.—Mercury, when quite pure, seems not to possess so great a power of taking up other metals, or of decomposing mineral combinations, as when it holds a minute quantity of some foreign metal in solution. The experience among amalgamators in Mexico is that the yield of gold is increased by the presence of silver; also, that the latter metal is extracted with greater facility if a considerable proportion of the amalgam is already present. This opinion is held by most mill-men in Washoe.

It is stated by some writers upon the question that silver is absorbed with increased activity when copper is employed, and as the former is amalgamated the latter will be expelled. Both iron and copper cause the formation of copper amalgam. On the other hand sulphate of copper exhibits a tendency to drive out lead.

Karsten mentions the property of this salt to purify the mercury from both zinc and antimony. Any one who has witnessed the intensity which sodium amalgam exerts cannot fail to have been impressed with the rapidity with which it attacks

gold, silver, and silver compounds; yet its application in Washoe in practical operations did not give such results as would warrant its general introduction in the process.

Although the presence of a small quantity of several metallic bodies enhances the amalgamating energy of the mercury, yet a slight excess "sickens" it; that is, it loses its fluidity and becomes dull and inactive. The peculiar phenomena attending the mercury, by which both electro-positive and electro-negative metals are absorbed, and the effects which they produce in increasing or neutralising its action, are very little understood.

The loss in quicksilver during the operation arises from two sources, the one mechanical, the other chemical. The former depends largely upon the manner in which the final washing from the pulp is conducted; the separation being more or less perfect according to the skill and care with which it is executed. A considerable quantity of the metal, however, is so cut up and ground to such a fine state of division that it is impossible to save it. The chemical loss is occasioned by the formation of the chlorides of mercury which escape with the tailings.

In the Washoe process the chemical loss would seem to be small in proportion to the entire consumption. This is probably due to the beneficial effects of the iron, which combines with the chlorine of the calomel, setting the quicksilver free.

Practical Conclusions.—From the foregoing considerations of the principal features of the Washoe process, and the experiments made by Mr. Hague, he draws the following conclusions :—*

1. That the ore consists chiefly of native gold, native silver, and argentiferous sulphurets, associated with varying proportions of blende and galena.

2. That the action of chloride of sodium and sulphate of copper in the pan produces chloride of copper.

* Report of Mr. Clarence King on the Mining Industries on the Pacific Coast.

3. That the presence of metallic iron necessarily causes the formation of the subchloride of copper.

4. That both the chlorides of copper assist in the reduction of the ore by chloridising in sulphurets of silver, and in decomposing the sulphurets of lead and zinc.

5. That sulphate of copper enhances the amalgamating energy of mercury by causing the formation of a small quantity of copper amalgam. It also tends to expel the lead.

6. That notwithstanding the importance of chemical agents, as above indicated, the quantities added to the pulp in the ordinary practice of Washoe mills are too small to effect any very beneficial result.

7. That mercury and iron, aided by heat and friction, are the principal agents in the extraction of the precious metals in the Washoe process.

8. That the essential conditions in the amalgamation of the gold and silver ore are, that the mercury be kept perfectly bright and pure, in order to produce a direct contact of that metal with the iron and sulphide of silver.

9. That the consumption of mercury in the Washoe process may be considered chiefly a mechanical and, only to a limited extent, a chemical loss.

CHAPTER VIII.

THE DRY PROCESS: CHLORIDISING-ROASTING, AND AMALGAMATION OF ROASTED ORES.

Rebellious Silver Ores—Operation of Roasting—Treatment of First-class Ores on the Comstock by Barrel Amalgamation—Treatment of Silver Ores at Mineral Hill—Introduction of Mechanical Roasters—Ore Elevator—Amalgamation of the Roasted Ore—Oakes Quicksilver Strainer—Melting and Assaying—Change to Wet Process—Taylor Mill at Mineral Hill—Treatment of Tailings from the Roasted Ore—Sampling—Working Ores at the Ontario Mine—At the Silver King Mine.

Rebellious Silver Ores.—Such ores as cannot be treated by the Washoe process on account of the quantity of base metals they contain have to be roasted before being subjected to amalgamation. The object of the roasting is not only to drive away the sulphur, antimony, arsenic, and other volatile products, and set the silver free from these combinations, but by the addition of salt to convert the base metals as well as the silver into chlorides, which, in the subsequent amalgamation, are reduced and combine with the quicksilver to form an amalgam. If the quantity of base metals, like copper and lead, be large in the ore treated, the bullion produced will be very base; and if the percentage of lead reaches a certain point, it renders the ore unfit for amalgamation altogether, and the ore has to be smelted or treated by the leaching process.

The dry process was first practised in Washoe on the rich ores, where they were roasted in reverberatory furnaces and treated by the Freiberg barrel amalgamation. Reverberatory furnaces are not much used in modern establishments, but we find them still in vogue in Mexico and some other countries. The barrel process also has nearly disappeared, but to keep up

the records I shall describe it in this chapter. In some European establishments it is still used.

The Operation of Roasting.—In the course of this operation, under the action of heat the sulphur takes fire, and burning away goes off as sulphurous acid or oxidized sulphur: and during the first stage of the roasting some of the oxidized sulphur will combine with some of the metals which under the influence of heat and air have been converted into oxides to form metal sulphates. Silver also is converted into a sulphate, or reduced into a metallic condition. Gold, if present, remains in a metallic condition. Antimony and arsenic are oxidized and go up the chimney, but a certain proportion may combine with some metallic oxides, forming antimonates and arsenates. Therefore the first stage of the roasting operation is termed *Oxidizing Roasting*, and we see that the oxygen in the air plays an important part in this operation.

When it is noticed that the evolution of gases and vapours decreases, the heat of the furnace is increased, the working doors of the furnace can be closed, as not so much oxygen is needed, and a decomposition of the metal sulphates will be effected. Sulphur oxide will continue to burn away, some metal oxides will volatilize, and such metals as iron and copper will remain behind as oxides, whereas some silver sulphate and lead sulphate remains, as the decomposition of these metals requires a higher temperature than is prudent to give, as it may cause a partial fusion or clinking of the ore. This second stage, or the reduction to oxide, is called *Dead Roasting.* This second operation is now very seldom employed in silver metallurgy, but is replaced by the more important manipulation called *Chloridizing Roasting.* By the addition of salt to the ore the chlorine is liberated through the action of the sulphates, and combines with the metals in the ores, forming chlorides, whereas the sodium in the salt combines with sulphur and oxygen to form sodium sulphate. Therefore when roasting ores in reverberatory furnaces it is always judicious to add the salt during the second stage of the roasting operation.*

* See some further observations on the subject at p. 344.

Some of the base metal chlorides volatilize, and when they take up oxygen become oxychlorides, or they give up their chlorine and become oxides, but the silver when converted in the furnace into a chloride remains as such. Some gold is also converted into a chloride. The metal chlorides which volatilize easily are antimony and arsenic; zinc volatilizes partly, so does lead. Through the volatilization of these base metal chlorides some silver chloride is generally carried along in the flues, although by itself the silver chloride is not very volatile.

Treatment of First-class Ores by Barrel Amalgamation.*—The establishment on the Comstock which was built for this purpose has twenty stamps for dry crushing, eight reverberatory roasting furnaces, and twelve barrels, capable of treating 300 to 400 tons per month. The principal features of the process will be briefly described.

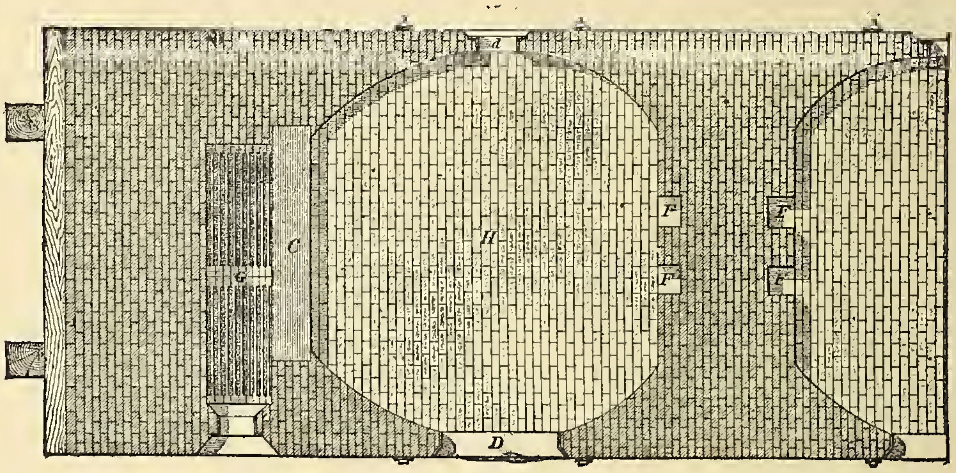
Drying.—The drying kiln consists of a series of flues, covered by a cast-iron floor, on which the ore, already reduced to a size suitable for stamping, is spread. The surface for the reception of the ore is about 8 ft. wide by 12 ft. long. The iron is cast in sections or plates, 8 ft. long by 3 ft. wide, with a strengthening rib on the under side. The base of the kiln is brickwork, and the flues are about 8 in. deep. They are covered by the iron plates. At one end of the kiln is a fireplace, and at the other a stack, so that the heat passes from one end to the other under the iron cover or floor, on which the ore is spread to a depth of 4 in. or 5 in. The ore is constantly raked and turned until quite dry. When the kiln is conveniently placed, as in some establishments in Eastern Nevada, the heat from the roasting furnaces, on its way to the stack, passes through the flues, saving a special firing. In the present instance there are three kilns, able to dry about 25 tons per day, consuming in all about half a cord of wood in twenty-

* Barrel amalgamation on the Comstock was practised in the early days of the operations there, and before I went to Nevada, and was described by Mr. Clarence King in his Geological Report of the Survey of the Fortieth Parallel, to which I am indebted for the matter under this heading.

four hours, and requiring one man's attention to keep up fires and rake over the ore.

Crushing.—For crushing the rock after drying there are twenty stamps arranged in batteries of four, weighing about 600 lbs. each, dropping 8 in. or 9 in. about 65 times per minute. The foundation and battery frame are not essentially different from those in wet-crushing batteries. The mortars differ from the high ones used for wet-crushing, consisting of a bed piece with sides and ends that are only high enough to provide the means of bolting the iron casting to the woodwork of the battery frame, attaching the screen frames, &c., &c.

The dies are flat circular pieces of cast iron that fit into recesses in the bottom of the mortar. Each die has two lugs or projections on its periphery, which, being dropped into a



Scale $\frac{1}{96}$.

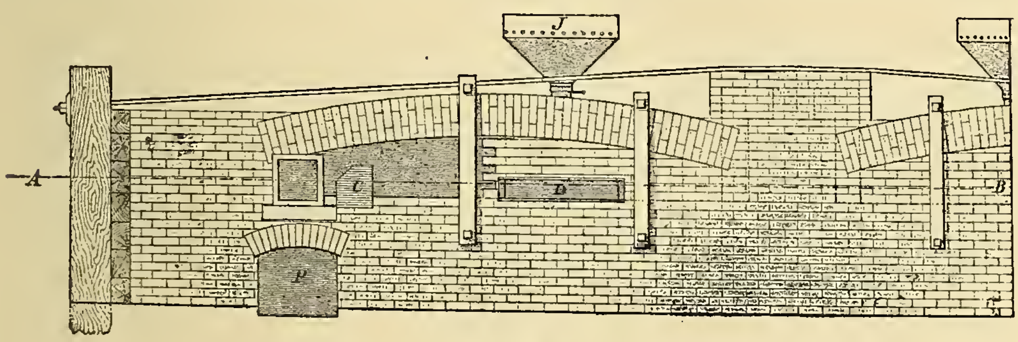
Fig. 46.—Plan of Roasting Furnace on Line A B.

groove in the bottom of the mortar, may then be revolved 90°, under a flange or lip with which the recess is cast. Molten lead is poured in to hold the dies firmly. When it is desired to remove them, quicksilver is poured into the battery, dissolving the lead and loosening the dies. By retorting the quicksilver both metals are recovered.

The discharge is at both sides and ends. Screens of brass wire cloth are used, having 40 meshes to the lineal inch, or 1,600 holes to the square inch. The stamps crush from half a ton to a ton per head per day of twenty-four hours. The bat-

teries are inclosed by housings or closely fitted boxes, which serve as receivers for the crushed material. The casings are provided with doors, by means of which the workmen can enter and remove the crushed ore by shovelling it into barrows.

Roasting.—The fine ore is then roasted with salt in reverberatory furnaces. These are built of common red brick. Figs. 46 and 47 show the method of their construction. Fig. 46 is a horizontal section through the line A B on Fig. 47. In the drawings H is the hearth; D, the stirring door; *d*, the discharge door; G, the grate; C, the bridge; F, the flues; P, the ashpit; J, the hopper. The charge consists of 1,000 lbs. of ore, which is mixed with 6 per cent. of salt, the latter being added to the charge in the hopper by which the furnace is



Scale $\frac{1}{96}$.

Fig. 47.—Elevation of Roasting Furnace.

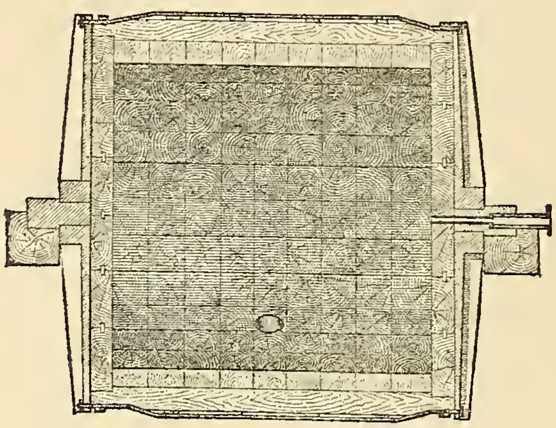
supplied. The charge is heated very gently at first, the temperature being gradually raised, until at the end it is subjected to a high heat. Usually six hours are required for the roasting. The charge is constantly stirred, and once or twice during the operation it is turned; that is, the portion of the charge remote from the bridge is caused to exchange places with that which is near.

The operation effected by this roasting with salt consists, very briefly expressed, first, in the oxidation of the metallic compounds, converting the sulphurets, in which form the silver chiefly exists in the ore, to sulphates; and the subsequent decomposition of these combinations by the salt, with the formation of the chlorides of the metals. Sometimes an addition of limestone is made to the charge, for the purpose of decom-

posing the chlorides of copper, zinc, &c., thus preventing to some extent their subsequent amalgamation in the barrel, and obtaining bullion of a purer quality.

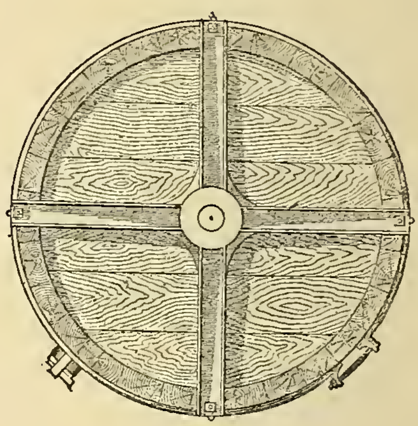
Each furnace, roasting four charges of 1,000 lbs. each, or 2 tons, in twenty-four hours, consumes one cord of wood. Two stirrers are employed on each twelve-hour shift, making four men in twenty-four hours. One man is required to receive and attend to the ore on the cooling floor after its discharge. The same man can attend to more than one furnace.

The roasted ore is passed again through a screen having 1,600 holes to the square inch, in order to remove from it any



Scale $\frac{1}{48}$.

FIG. 48.—FREIBURG BARREL.



Scale $\frac{1}{48}$.

FIG. 49.—END VIEW, FREIBURG BARREL.

lumps that may have formed by caking in the furnace, or coarse particles that may have escaped the battery screen. It is then elevated to a large copper, placed above the amalgamating barrels, to which latter it is thence supplied by means of smaller hoppers, one of which is suspended over each barrel.

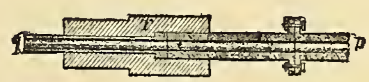
The reactions during the roasting are: when the heat is sufficiently high to burn the sulphur, dense greyish white vapours of arsenic and antimony are exhaled if the ore contains these combinations; a blue flame indicates the commencement of desulphuration or oxidation, by which the sulphides and antimony are mostly transformed into salts, part of them also into free oxides. This continues for about two hours, during which time the ignition is kept up, and the mass is

thoroughly turned over in order to present new surfaces and prevent caking. When sulphurous acid ceases to be formed, the final calcination must be commenced with increased firing, the object now being to decompose the salt by means of the metallic sulphates which have been generated, and to convert them into chlorides, with the simultaneous production of sulphate of soda. The stirring is to be continued till the samples drawn from the hearth no longer smell of sulphur, but only of hydrochloric acid. The roasting mass will then have assumed a woolly appearance. This stage of the roasting lasts about one hour.

Amalgamation in the Barrel.—The barrels are four or five feet in length and diameter. They are usually made of soft pine. Figs. 48 and 49 show a vertical section and end view of an

Scale $\frac{1}{24}$.

FIG. 50.—IRON HOOP.



Scale $\frac{1}{24}$.

FIG. 51.—STEAM PIPE.

amalgamating barrel formerly used at the Gould and Curry mill. The ends of the barrel are made of plank nicely fitted together and joined with a tongue of hard wood. The staves of the barrels are sometimes made of six-inch stuff, without lining; sometimes, as shown in the figure, the staves are two or three inches thick, with an interior lining of blocks four or five inches square, and three or four inches thick, and so placed in the barrel that the wear is on the end of the grain. This lining can be removed when worn out. The staves of the barrels are bound with iron hoops, the ends of which are drawn together as shown in Fig 50. The ends of the barrel are strengthened by a four-armed flange of cast iron. The barrels are caused to revolve by cog gearing, the teeth being put on in segments around the end of the barrel, or by belting, or by friction gear. The barrel, of which a section is seen in Fig. 48, shows a contrivance for admitting steam to

the pulp through the trunnion. This arrangement, not very common, consists of a steam pipe, *p*, Fig. 51, which enters the trunnion and fits smoothly against the end of another pipe *q*, that passes through the end of the barrel and admits the steam to the interior. The interior pipe, *q*, revolves with the trunnion, while the exterior pipe, *p*, is fixed and remains without motion. The trunnion, T, is keyed to the flange already referred to.

The barrels are charged with about two thousand pounds of ore mixed with water enough to make a moderately thick paste. Before adding quicksilver, the charge is revolved for two or three hours in the barrel with several hundred pounds of scrap iron. The object of this is to effect a partial reduction of the chlorides present, which would otherwise be performed at the expense of the quicksilver. The chloride of silver is partly reduced by the metallic iron, and is subsequently amalgamated by the quicksilver. The same is true of the lead and copper. Quicksilver is added according to the richness of the ore, usually varying from 250 to 500 or more pounds. The barrel is run two hours, at twelve or fifteen revolutions per minute, and then examined, that the consistency of the paste may be ascertained. If the latter is too thin, the quicksilver settles on the bottom. This condition is remedied by the addition of more roasted ore; while if too thick for the most favourable distribution of the quicksilver, more water is added. The barrel is then allowed to revolve again for fourteen hours, making fifteen revolutions per minute. The whole time occupied from the charging to the discharging of the barrel is eighteen or twenty hours. When the amalgamation is complete the paste is thinned by the addition of water, and the quicksilver and amalgam are thus allowed to collect on the bottom of the barrel.

Below the barrels is a large hopper or funnel-shaped contrivance, sloping down from the four sides to a common centre. When a barrel is to be discharged a small plug in the side is loosened while turned upward, and when the barrel is revolved so that the plug is downward, it is drawn out by hand. The quicksilver and amalgam are discharged into the hopper or

funnel just described, and are allowed to run from the barrel until the pulp begins to follow, when the plug is replaced. When all the barrels ready for that purpose are discharged, the amalgam in the hopper is carefully collected and washed and afterwards cleaned in a common pan, like those in use in other mills for similar purposes. The straining of the quicksilver and retorting of the amalgam is performed in a manner similar to that already described.

After the hopper below the barrels has been cleaned of all the quicksilver discharged into it, the residue is permitted to flow from the barrels and to run down into a large agitator 8 or 10 ft. deep, and 12 or 15 ft. in diameter, in which stirring arms are revolving in a manner similar to that already described. By this means the unseparated quicksilver and amalgam are allowed to settle, and the concentrations of this vessel are worked over in pans in the common way, while the mass of tailings passing from the agitator are subjected to further methods of concentration and subsequent treatment.

Chemical Reaction in the Barrel Amalgamation Process.—The metallic chlorides present in the roasted ore are decomposed by the iron, whence results chloride of iron, whilst the chloride of copper is reduced partly to subchloride and partly to metallic copper, which then precipitates metallic silver. The mercury dissolves the silver, copper, lead, antimony, &c., forming a complex amalgam. If the iron is not present in sufficient quantity, or if it has not been worked with the ore long enough to convert the chloride of copper into subchloride previous to the addition of the mercury, more or less mercury will be wasted by its conversion into calomel. Chlorides of manganese, zinc, nickel, and cobalt are not transformed, and may be afterwards precipitated from the solution by lime; if many foreign chlorides are contained in the mass in consequence of too low a temperature during the roasting process, some lime may be added to decompose them. When the ores are very rich in copper, copper is added instead of iron, as iron causes the formation of an amalgam rich in copper.

Treatment of Silver Ores at Mineral Hill, Nevada.—The experience gained in the employment of reverberatory furnaces on the Pacific coast proved that they were unsuitable to the local requirements and conditions. They were expensive to operate, and required not only skilled but reliable labourers, who are rarely to be obtained in the Far West. Moreover, the output was too small where large quantities of ore had to be regularly treated. The invention of the mechanical roasters overcame all these difficulties, and they are now almost exclusively used. They require labourers of no particular skill, and the whole manipulation of the ore in the mill becomes simplified from the fact that from the time the ore is charged into the battery no handling to any extent is requisite, as everything is done automatically by machinery. I shall give a description of several of these furnaces, which all have more or less their particular merits, but those mostly used are the Stetefeldt furnace and the Brückner roasting cylinder. They are divided into two classes—namely, shaft furnaces of the Stetefeldt type, named after the inventor, and revolving cylinders.

In the following pages I shall describe the operation of the dry process as first carried out at Mineral Hill by myself. I used the roasting furnaces, however, for a very short time only, as I found that I could treat the ores to much better advantage by the Washoe process.

The ores I dealt with at Mineral Hill occurred in irregular masses in a shallow limestone formation, which formed the crest of a promontory of an irregularly stratified slate. The ore bodies were large, but were unfortunately only confined to the limestone capping, and did not, as was supposed at the discovery of the mines, penetrate into the slate or form a contact deposit between the two formations. They were of somewhat complex nature, containing chloride of silver, bromide of silver, argentite, polybasite, stephanite, carbonate and molybdate of lead, carbonate of copper, and some manganese. The first discoverers of the mines took eighty tons to Austin for treatment in the Manhatten mill, where the ore, after roasting, gave by pan amalgamating excellent results. It was concluded that the ore required

roasting, and in consequence a 15-stamp mill was erected at Mineral Hill, provided with a Stetefeldt furnace. When the property was sold to the London company and I took charge of the reduction works, another mill of 20 stamps, likewise provided with a Stetefeldt furnace, was also erected.

The operation in the 15-stamp mill was as follows: The ore as brought from the mine was delivered at the Blake rock-breaker, and after being crushed into small pieces fell directly on to the drying floors. As it then contained about 4 per cent. of water, it was in that state unfit to be passed through the dry crushing battery. The drying-kiln, constructed as has been previously described (on page 147), was heated from a special furnace placed at one end. The salt was always dried on the same floor, but separately, and mixed afterwards with the ore.

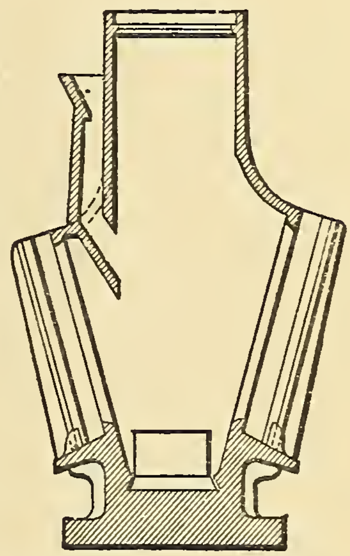


Fig. 52.—Dry Crushing Mortar. Double Discharge.

After drying, the ore was shovelled on to the space in rear of the battery and fed by hand labour into the battery, one man working twelve hours attending to the fifteen stamps. He was relieved twice on a shift during one hour by some of the other workmen to take his refreshments, the night shift releasing the day shift at 6 P.M. throughout the whole mill.

The battery was similar to the ordinary battery and provided with screens of brass wire cloth, with 40 meshes to the running inch. In dry crushing mills the discharge ought to be lower than in wet crushing, as the ore is simply thrown by the fall of the stamp against the screen, and only passes through the screen by the impetus imparted to it through the drop of the stamps. The stamps weighed 850 lbs., 63 drops per minute with 8½-in. drop. The mortars discharged on one side (not on both sides as shown in Fig. 52), and were completely housed in; and on discharging the fine pulp fell into a launder in which moved an archimedean screw, carrying the

pulp to a point situated at the end of the battery, where it fell into a box and was raised from there by the "ore elevator," shown in Fig. 53, to the bins placed above the feeding apparatus of the Stetefeldt roasting furnace.

Ore Elevator.—This apparatus, which secures a safe automatic conveyance of the pulp to the roasting furnace, is composed of a series of sheet-iron cups, attached at regular intervals to an endless belt passing over pulleys; and by its means the crushed pulp from the battery is delivered as fast as received to the hoppers overhead. It is surrounded by a wooden casing to prevent escape of pulp into the mill.

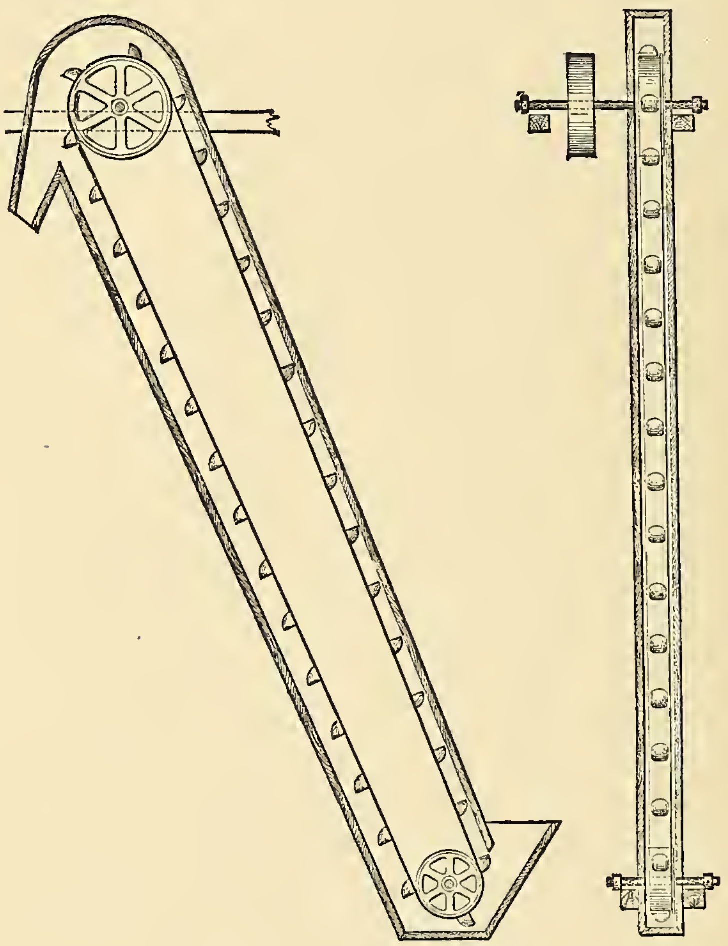


Fig. 53.—Ore Elevator.

The housing-in of the battery, launder with archimedean screw, and ore elevator, should be done very snugly, so that no dusting may take place, as the dry dust floating about not only interferes with the working parts of machinery and causes loss of metal, but is highly injurious to the health of the workmen, as will be readily understood when men have to inhale dust containing lead, antimony, and in most cases arsenic. It is, therefore, advisable to attach an exhaust fan to the housing, and to carry the dust to a point above the roasting furnace.

After drying I mixed the salt together with the ore, using

from 3 to 5 per cent. according to the character of the ore, and when the pulp reached the hopper it was fed therefrom by a very ingenious contrivance into the roasting furnace.

This furnace—the Stetefeldt—I will (for sake of convenience) describe in detail in the next chapter, which deals with various descriptions of roasting furnaces. There also will be found an account of the operation of roasting and the subsequent cooling of the ore, preparatory to the next stage of the process, as followed by me at Mineral Hill.

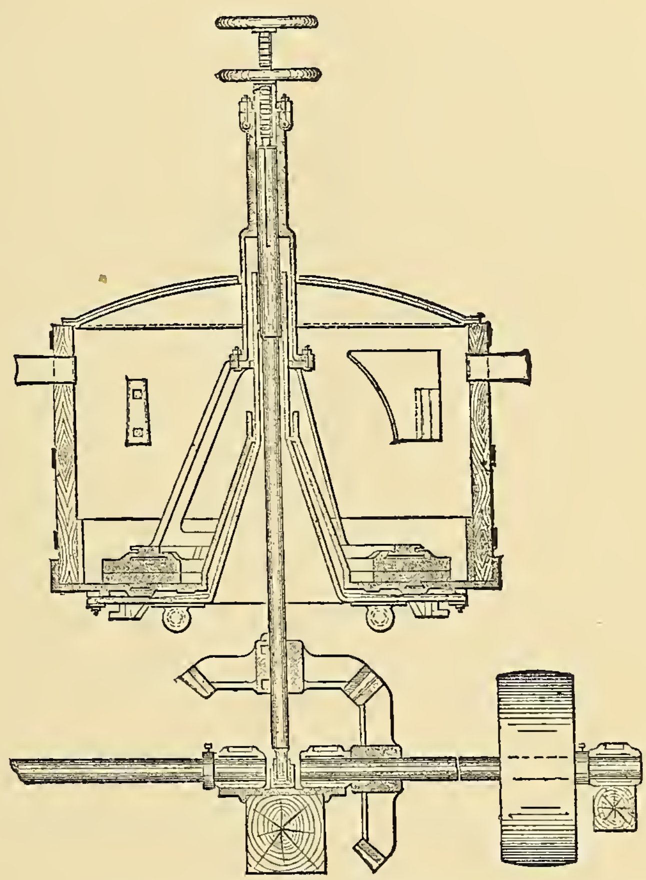


Fig. 54.—Pan with Wooden Sides.

The Amalgamation of the Roasted Ore.—From the cooling floor the pulp would be carried on a tram-car into the adjoining pan-room. We had eight combination pans of the Wheeler pattern, which discharged into four settlers. The pan-room was so arranged as to have four pans and two settlers on each side, with the tramway in the centre. Each pan was charged with 1,500 lbs. of ore, sufficient water added to give it the proper consistency, and a dipper full of sulphuric acid also added.

The ores contained a certain proportion of copper, and I found that the addition of the acid slightly increased the results in amalgamation. Whenever the ore contained plenty of copper the percentage of silver extracted was higher, reaching sometimes 92 to 93 per cent., and the lower the fineness of the bullion produced, the higher was the percentage of silver extracted from the ore, showing that copper in the ore assists

the reaction in pan amalgamation. I did not grind, but simply ran the mullers for two hours before adding the quicksilver, keeping up steam in the jackets. After adding from 300 to 400 lbs. of quicksilver to each pan, and adding 10 to 15 lbs. of iron borings and scrapings purchased from machine shops in San Francisco, I ran the same for six hours longer, and then discharged into the settler. I believe in the addition of plenty of quicksilver, which should be kept clean and active —a rather difficult task when treating ores containing lead.

The pans were built up with wooden sides on a cast-iron base, similar to the pan shown in Fig. 54, and subsequent experience has shown me that similar pans can be used to equal advantage in working raw ores by the wet process. I question the advantage of using iron pans in the amalgamation of roasted ores, as the corrosion of iron is very large, and mullers, shoes, and dies are eaten away rapidly, especially when treating rich ores. I regret that I have not the records of the consumption of iron in the pans, but it was very large, and probably pans built up of stone like arastras, with iron borings as reducing agent, would prove very economical in distant mining regions where the freight of iron is expensive. The settlers discharged into launders, carrying the tailings to the tailing pit. I ran the settlers two hours before commencing the discharge at the top hole. The chloride of silver, and also some lead and copper chlorides, were reduced by the iron of the pan, and no doubt also by the subsequent addition of the mercury; but as the loss of mercury did not exceed 1½ lb. per ton of ore, and these ores assayed from 100 to 130 oz., I believe that the quantity of mercury lost by chemical action in this process is small, and I am certain that the loss is mostly a mechanical one, mercury being such an unstable metal that losses cannot be avoided in its manipulation. The amalgam was gathered in the usual manner and retorted.

The H. H. Oakes Quicksilver Strainer * was introduced

* Mr. Oakes was the Managing Director of the Mineral Hill Mines, and this apparatus was introduced at his suggestion. It proved very effective.

in the mill for the purpose of cleaning the quicksilver from the impurities which became mixed with it during the process of amalgamation.

When the quicksilver is strained from the amalgam canvas bag it drops into a tub, together with dirty water, fine pulp, and greasy matter, and these impurities are usually removed from the surface of the quicksilver by means of a sponge, and of course globules of quicksilver remain sticking to the sponge, and mechanical loss is unavoidable during the manipulation.

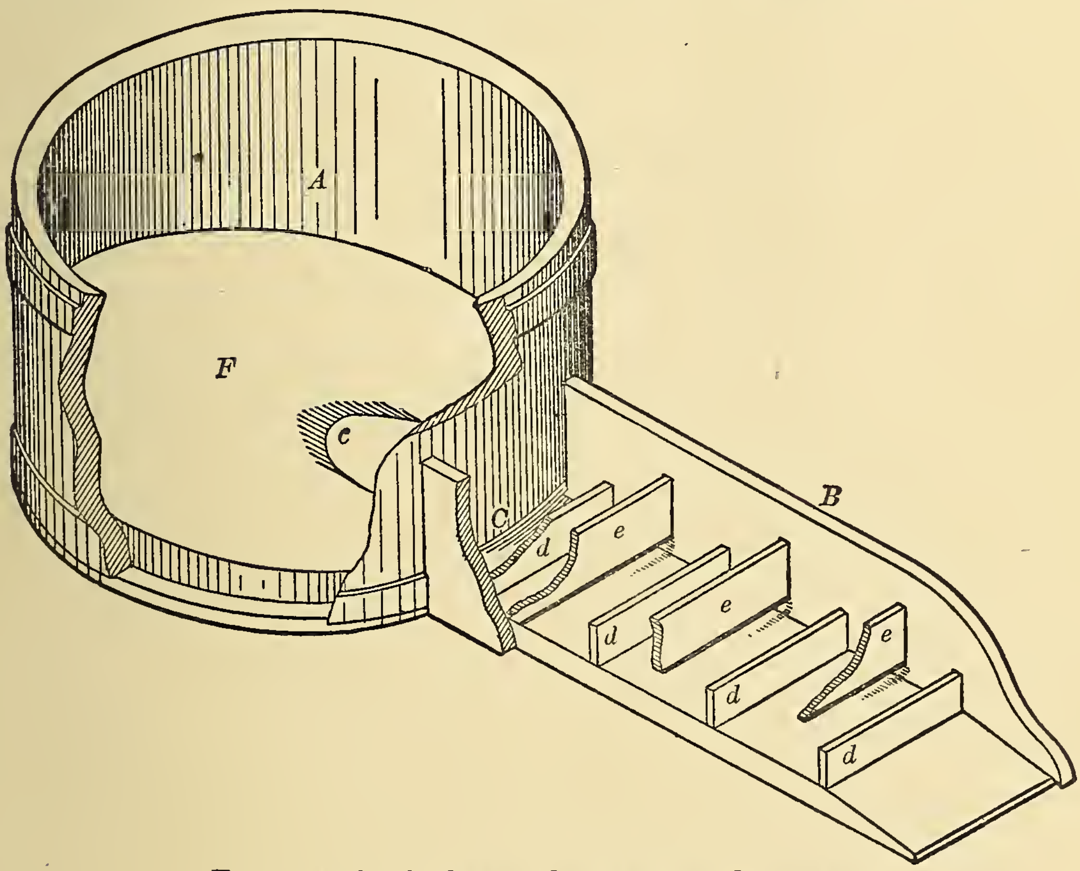


FIG. 55.—H. H. OAKES QUICKSILVER STRAINER.

The Oakes apparatus, of which Fig. 55 is a perspective view and Fig. 56 a section, does away with the handling of the quicksilver, which comes out clear and bright for the next charge in the amalgamating pan. A represents a tub, into which the quicksilver drips from the amalgam sack. To one side of this tub the spout, B, made of iron, is secured. The bottom of the spout, B, is on a level with the bottom of the vessel, and a small orifice, C, is made through the side of the vessel on a level with both bottoms. This orifice is made as

wide as the spout and quite narrow. Across the spout, B, are arranged several riffles, *d e*, which are wooden partitions. The riffles, *d*, extend only a short distance upward from the bottom of the spout, their lower edges being fitted closely against the bottom, while the partitions *e*, which alternate with the partitions *d*, extend from the top to within a short distance of the bottom of the spout, so as to leave a narrow space between them.

By these means the quicksilver which enters the spout through the slot or orifice, C, will be compelled to pass alternately over the partitions or riffles, *d*, and under the partitions, *e*, in order to flow through the spout.

A false bottom, F, the upper surface of which inclines towards

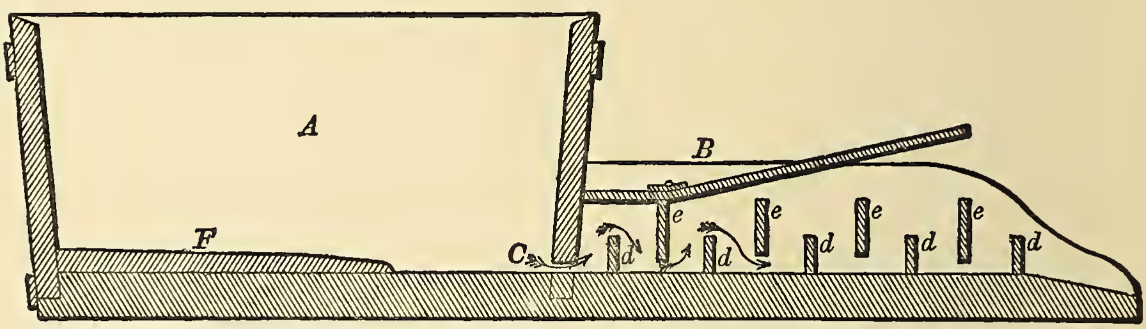


FIG. 56.—H. H. OAKES QUICKSILVER STRAINER.

the orifice, C, is placed in the bottom of the vessel, A, so as to carry the quicksilver to the orifice and direct it into the spout. The quicksilver in passing over and under the riffles leaves the impurities behind and comes out bright from the spout.

Melting and Assaying.—The amalgam averaged 200 lbs. silver to 1,300 lbs. amalgam. It was retorted and melted in a combination retort and melting furnace, as shown in Fig. 57. When melted, the bars, which I generally made about a hundredweight, contained from 600 to 740 parts of silver, the balance mostly copper with some lead. After making a fire assay of the ingots, I assayed them by the Guy-Lussac volumetric method, to be described hereafter.

To control the output of the mill, every half-hour samples

were taken of the tailings discharging from the agitators. A perfect record was kept of the quantity of ore passed through every twenty-four hours; and the average assays of the day and night samples of ore and tailings, and the bullion production at the end of the month, had to correspond with the resulting figures. It may be a matter of surprise that, in spite of the greatest care in my methods of calculating, I had a plus bullion production at the end of the month, exceeding by £200 to £250 the amount my books required to be produced by the

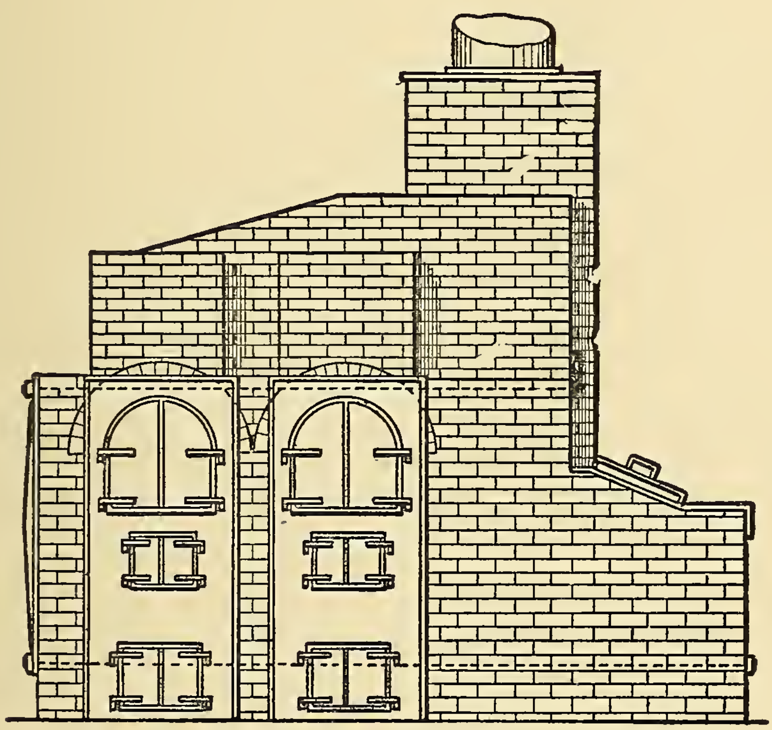


FIG. 57.—RETORT AND MELTING FURNACE.

mill, which is evidence that the volatilization of silver in the Stetefeldt furnace was nil.

The silver bullion which we produced was sent to San Francisco, and the bank deducted 2 to 2½ per cent. of its assay value, on account of the large percentage of copper it contained, to cover the cost of refinage. As the wear of iron in the pans was considerable, and to avoid the destruction of the mullers, shoes, &c., we added iron shavings; but still we could calculate the average wear throughout the mill at 15 lbs. of iron per ton of ore treated.

Change to the Wet Process.—I manipulated the mill in this manner for a few months, when one day I concluded to make a chlorination test of the raw pulp, to see what percentage of free chloride of silver the ore contained, when, to my great astonishment, I found it to amount to 46 per cent. I at once set to work experimenting in the mill, and commenced working the ores raw, with the result that I closed down the Stetefeldt furnace and adopted the Washoe process. I worked the ores up to 90, and on some days to 92 per cent., and the bullion which I produced by this method was from 920 to 970 fine, and instead of the bullion being discounted, we got a premium of 1½ per cent. from the bank. The discarding of the Stetefeldt furnace and expensive roasting not only saved the company £1,000 per month, but allowed us subsequently to work many thousands of tons of low grade ores, which, when I took charge of the mill, were considered worthless, as they would not have covered the cost of treatment by the roasting process, but which by the Washoe process yielded a profit.

I charged, as before, 1,500 lbs. of ore with from 15 to 20 lbs. of salt, and from 3 to 5 lbs. of bluestone, which after many experiments proved to be the best proportions. I ground the ore in the manner described under the Washoe process.

The chief influence of salt is to form chloride of silver from the decomposition of the sulphides; and this is very desirable, as this salt is so easily decomposed by copper, iron, and mercury. The sulphate of copper produces, with the copper present in the ore and with the salt, chloride of copper, which by the iron in the pans is partly reduced to subchloride; these chlorides act on zinc-blende and galena, and prevent them from being absorbed by the amalgam. The sulphate of copper—and when the ores carried much copper I also added sulphuric acid—acted on the lead ores and converted them into insoluble sulphate, and the metallic copper set free amalgamates with the mercury. The presence of binoxide of manganese always reduced the percentage of the silver production, as it seemed to flour the quicksilver. Its presence was always indicated in the

settlers by a thick froth, which in spite of dilution with water would carry off floured quicksilver. I know of no remedy to counteract this influence.

The pans never stopped when there was ore to treat, except for repairs or a clean-up. Each pan was arranged so that it could be stopped when necessary independently of the others. The amount of wear of a pan depends on the character of the ore to be ground, but the body of a pan should last several years; and as I obtained such good results by the Washoe process in pans with wooden sides, which are more economical, especially in countries where cost of transportation is expensive, I see no reason why this form of pan should not be universally adopted.

I crushed the ore dry, as by wet crushing too large a percentage of chloride of silver would have floated away. The daily capacity of the mill was 18 tons.

Cost of Treatment.—The cost of the method was as follows in daily outlay:—

Superintendent, who also acted as assayer . . .	\$10
Master mechanic	6
Carpenter	6
Two engineers, at \$5, one day and one night . . .	10
Two men to receive the rock and attend Blake rock-breaker, at \$4 per day	8
One man to attend dry kiln and take battery sample, one night and one day	8
Two battery feeders, one day and one night, at \$4½ . .	9
Two pan men in amalgamating room during the day .	8
Two ditto, and one retorter at night	12
400 lbs. of salt at 6 cents	24
8 cords of wood at \$6	48
Loss of quicksilver, 30 lbs. at \$1·20	36
Wear and tear of iron and repairs	20
Oil and incidentals, and sulphate of copper and assay material	15
Cost of treating 18 tons per day . .	\$220

Or \$12.22 per ton.

Taylor Mill at Mineral Hill.—Fig. 58 shows a section of the second mill, or Taylor Mill, of 20 stamps, which was only operated for a few months, the supply of ore falling

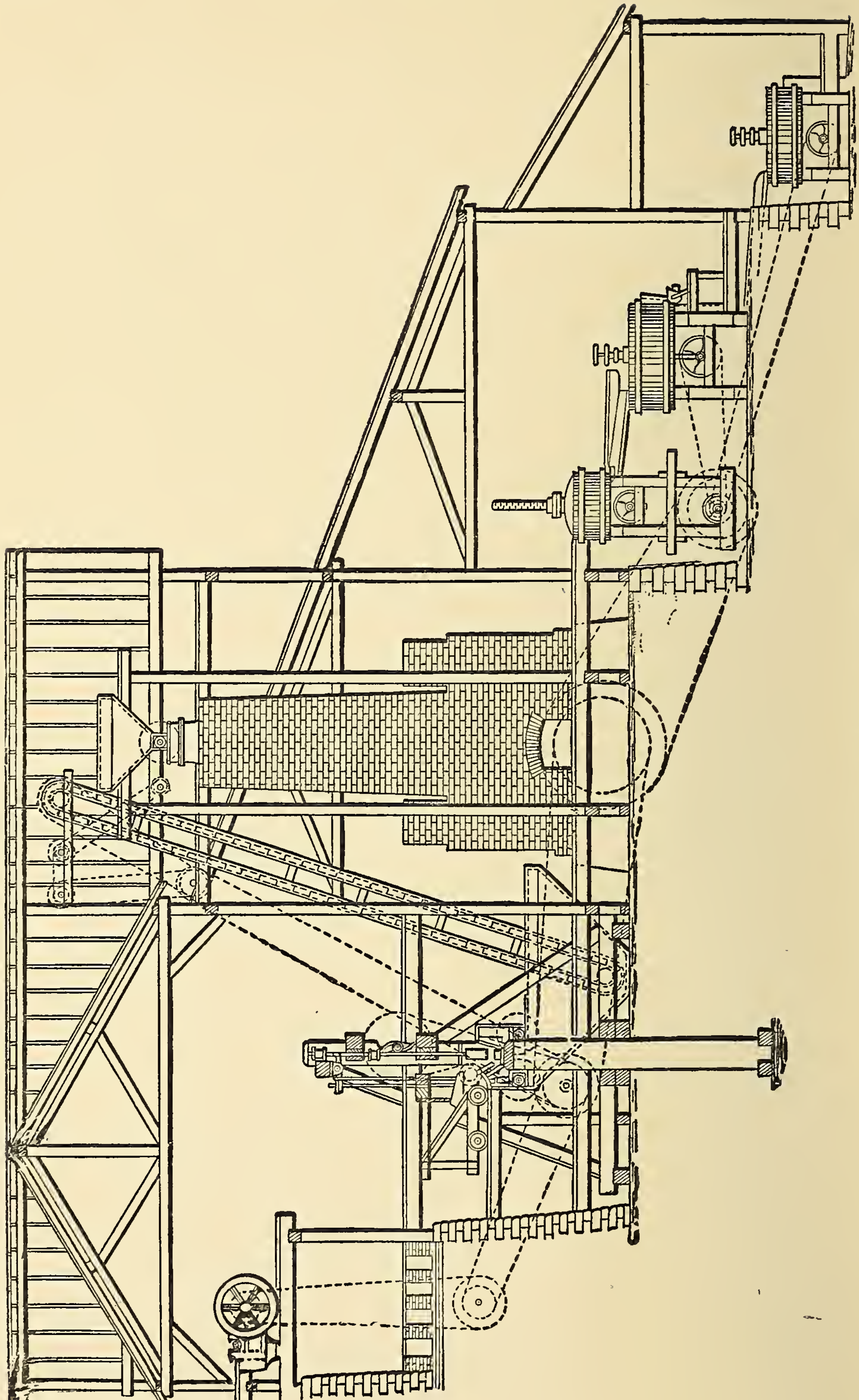


Fig. 58.—The Taylor Mill at Mineral Hill. Scale $\frac{1}{200}$.

short. This mill only differed from the other in that it was provided with Stanford self-feeders, 12 pans, 6 settlers, and 3 agitators.

I employed the pans of this mill during several months in the treatment of the roasted tailings, which had accumulated to several thousand tons in the tailing pit, from the previous operations.

Treatment of Tailings from the Roasted Ore.— After careful sampling of the tailings pit, I found them to contain on an average 16 dols. silver per ton; and as they had been exposed to the influence of atmospheric agencies for over two years, the decomposition of some of the unreduced sulphides had been effected, and I arranged the pans of the Taylor mill for their treatment. I put in new shoes and dies, so as to grind them and submit them to the Washoe process. The first charge being drawn, everybody was astounded at the large quantity of amalgam which the strainers gave us; but it was not the crisp, hard-grained silver amalgam, but a greasy dull-grey substance, indicating the presence of base metals. I at once filled one retort with the stuff, and after the operation was finished I withdrew a soft black metal, which I cast into bars. These bars were lead ingots, containing 60 parts of silver, about 290 parts of copper, and the balance lead.

I commenced the operation by using 300 lbs. quicksilver per pan, which was added after the ore had been ground for two hours with 30 lbs. of salt and 3 lbs. of sulphate of copper, running the charge six hours; the tailings showing that a result of only 36 per cent. had been obtained for the first two days' working. I modified the process by adding 600 lbs. f quicksilver per charge, using 10 lbs. of salt and 5 lbs. of sulphate of copper, which increased the percentage to 58 per cent. of the assay value, and this by careful manipulation I raised to 62 per cent., the highest result I was able to obtain.

My greatest difficulty was to keep the quicksilver clean and to retort this base bullion, as it would "froth" in the retort and boil over the cups in the operation, and it was quite a tedious

job to withdraw it from the retort. To avoid this inconvenience, I arranged an iron kettle with steam-jacket, and the quicksilver as it came from the settlers was heated and strained in a hot condition. This would leave the silver amalgam in the bag, but the lead remained dissolved in the hot quicksilver, which would strain into a tub of cold water, and on cooling would be strained again, giving a lead amalgam. These amalgams, one called "white," the other "black," were retorted separately. The white gave a silver bullion from 360 to 480 fine, the balance mostly copper, with some lead; and the black gave lead ingots, still containing from 10 to 30 parts of silver, with some copper.

The 7,000 tons of tailings which I treated gave a profit of about 12s. per ton, or some £4,000 over and above all regular and incidental expenditure.

There was no loss of quicksilver during the whole operation, and when the general clean-up was made I found I had a slight excess in weight of quicksilver of the quantity I had when the treatment of the tailings was begun. This (I surmised) must have resulted from the regaining of the particles of quicksilver mechanically lost during the pan amalgamation of the ores, which were collected in the agitators during the treatment of the tailings—a gain which counterbalanced the chemical loss occasioned during the treatment of the tailings.

I charged the 12 pans with 1 ton of ore each four times a day, working 48 tons in twenty-four hours; and it was only owing to the great quantity treated that I was able to realise a profit.

This was the first instance in which tailings from roasted chloridized ores had been successfully and profitably treated in Nevada, and, as far as I know, on the Pacific Coast. There were at that time many wiseacres who predicted a failure when I tried my experiment, and prophesied that I would not obtain one ounce of silver from a ton of tailings; but in this—as in many other metallurgical operations—it is found that the precious metals can be extracted from the most refractory ores by applying suitable chemical agencies to effect their separation and subsequent precipitation, so that their combination with quicksilver may be effected.

Sampling.—The working of the mill was under perfect control by means of systematic sampling and assaying. The ore was first sampled every half-hour as the pulp discharged from the mortar, a box being held before each mortar for a few seconds, and every twelve hours this "battery sample" was delivered to me. On top of the furnace stack, the feeder took every half-hour a sample from the mouth of the hopper, where it fed into the furnace, and every twelve hours the "feeder's sample" was delivered. The furnace men on withdrawing charges of roasted ore with a sampling iron draw proper samples, which were collected in an iron vessel and also delivered to myself every twelve hours.

I made duplicate assays of each of these three samples, which gave me the assay value of the ore crushed during the past twelve hours, and the "roasted ore" of course indicated always a slightly higher result than the "raw ore." While conducting these assays I made simultaneously the "chlorination test" of the roasted sample. I took a few ounces of it on a filter, which I leached with hyposulphite of soda until all the silver that was in a soluble condition was dissolved, which required from twelve to twenty-four hours. The filter was burned and calcined with its residue, and an ordinary assay made of it; and as an assay of the unleached ore had been made, the amount of the silver leached out by the hyposulphite of soda ought to represent the choride of silver, and what remained in the residue was the amount of silver not converted into chloride, and of such silver salts as were not soluble in hyposulphite of sodium. It is a fallacy to think that whatever silver is leached out by the hyposulphite must be in the state of a chloride or a silver sulphate; but as I generally worked up in the amalgamation process over and above the chlorination test, I took it for granted that the chlorination was perfect whenever the same exceeded 90 per cent.

On an average 90 per cent. of the assay value of the ore was extracted. The ores contained no gold, and in the best times of the property they averaged £30 to £40 per ton, but on the decline of the mines I worked ores which only assayed £6 per ton.*

* For a description of Brunton's ore sampler, see *post*, p. 342.

Silver Milling Ores at the Ontario Mine, Utah.—One of the best examples of a silver mill is the mill of the Ontario Silver Mining Company in Utah, one of the most successful concerns in the West. I am indebted to a report of Mr. R. P. Rothwell* for the following account of its operations :—

"This mill treats the ore from the Ontario mine—ore which at present is very base, being composed of zinc, lead, and silver sulphides and silver chloride in a quartz gangue. This ore has become baser as the mine attained greater depth, though the vein holds its own, or rather increases, both in thickness and richness. The mill is managed in a skilful and economical manner, though the necessity for roasting the pulverised ores in Stetefeldt furnaces necessarily makes the cost of milling much greater than in the case of such free milling ores as those of the Comstock mines, and especially than those unrivalled free milling ores of the silver-bearing sandstone district of Silver Reef in Southern Utah.

"The data here given were furnished by Mr. T. C. Chambers, the general manager of the mine, and Mr. T. E. Gallagher, the superintendent of the Ontario mill.

"The mill is charged with the hauling of the ore from the mine to the mill, which costs by contract 50 cents per ton dry. The moisture averages about 11 per cent. This distance is about a mile, and down grade all the way, four-horse teams hauling six to seven tons at a load. The ore, which is weighed by one weigher at $4 a day, is delivered on the ore floor screened, by passing over iron bars 2 in. by $\frac{5}{8}$ in. by 9 ft. long, with spaces of two inches between; the inclination of the bars is about 30°. The coarse ore goes through two Blake crushers, attended by one man at $2½ per day, and is crushed to the size of a pigeon's egg, and then goes along with the fine ore which has passed between the screen bars through shoots to the drying floor, which is heated by thirteen flues, seven of which are heated by auxiliary fires and six by the waste gases from the Stetefeldt furnaces and one auxiliary fire.

"It was at first supposed that the waste gases from these

* Transactions of the Institute of American Mining Engineers.

roasting furnaces would have sufficed for the drying floor, but experience has shown that they do not. From the shoot the ore is carried over the drying floor in cars, which carry 1,000 lbs. of ore (dry), and from this measure the quantity of salt is gauged. The ore is spread over the floor to a depth of 3 in., and, after drying for two hours, 17½ per cent. wet (about 15 per cent. dry) of salt is scattered over the ore. The salt used is evaporated from the waters of Salt Lake, and costs at the Ontario mill about $7 per ton, $5½ of this being for hauling from the salines to the mill. The ore is left from one and a half to two hours after the addition of the salt for the purpose of drying the same, and is then turned with shovels. This work is very injurious to the men, who quickly get swelling of the legs, it is supposed from arsenic in the ore; and in this, as in most other trying occupations, it is found that indulgence in alcoholic stimulants quickly incapacitates the workmen. Three men on a shift and three shifts a day, with wages at $3 50 cents. per day, are required for the work of the drying floor. After turning, the ore and salt remains on the floor for from one half to one and a half hours, and is then scraped off by a mechanical scraper to the self-feeder of the battery.

"The battery consists of forty 800-pound stamps, which drop 8 to 8½ in. 92 times per minute. The shoes, dies, and tappets are of steel, cast in Collinsville, near Hartford, Conn. The dies and shoes wear from two to six, and sometimes eight months; the tappets and cams last twelve months. The cams weigh about 250 lbs., and the iron stamp stems are 3⅛ in. diameter and 14 ft. long. The stamp screens are of brass wire No. 30, that is, 900 meshes in a square inch. Formerly a No. 50 screen was used, but as the ore became more base it would not splash high enough on the screen, and the amount it was possible to mill dry became greatly reduced. Mr. Gallagher tried the larger mesh screens with great success, it being found that the roasting and chlorination in the Stetefeldt furnaces is quite as perfect, if not more so, on the coarse ore as on the fine. The battery requires the attendance of one man each shift; wages, $4 dollars, and three shifts

a day. The crushed ore is conveyed from the battery by a screw shaft, working in a box parallel with the battery to a common flour-mill elevator (with Russia iron cups 22 in. apart, on a rubber belt 6 in. wide, and which lasts two years). This raises it to a vertical height of about 50 ft., to the pockets over the Stetefeldt furnaces; thence it passes through two screens, the bottom one of which is fixed, and stands on the top of the water-jacket, and the upper one vibrates, making about 50 strokes per minute. It is run by cone pulleys, and over the vibratory screen are four brass bars called "agitators," which keep the ore moving on the screen. The screens are of No. 18 steel plate, punched with ten holes to the inch, and the ore sifts down through them into the furnace. This is 46 ft. 6 in. high, and from the bottom the ore is drawn every three-quarters of an hour. One man on a shift (wages $4), and three shifts a day, is all the labour required at the screens. This man takes samples for assay every hour.

"The furnaces have thirteen dust chambers—8 and 10 ft. high, 8 ft. long, and 4 and 5 ft. wide—which collect about 25 tons of dust per month; this averages a little higher grade than the ore; about 56 per cent. of it is soluble. The two furnaces have a capacity of 50 to 55 tons a day of the ore now being treated, and consume daily 9½ to 10 cords of wood, which costs $4 per cord. Firing requires one man (wages $4) on each shift, three shifts per day. After leaving the furnace the ore goes to the cooling floor, where it remains piled up for eighteen hours; this assists the chlorination about 8 per cent. on the ore from the old furnace, and 3 per cent. on that from the new. After this it is wet with water and run in cars to the pan-room. Three shifts of two men per shift, at $4 a day wages, attend to the cooling floor; two shifts, with two men on a shift, wages $4 a day, attend to the cars, and take samples for assay from every car.

"There are twenty-four pans, which are charged each with 2,500 lbs. (dry) of pulp and about 1 per cent. of salt, and the pulp is made, by the addition of hot water, into a paste of about the consistency of thin mortar. The muller is held about

an inch off the bottom, and consequently does not grind in the pan. It makes about 65 revolutions per minute and runs for eight hours. About 1 lb. of zinc, costing 9 or 10 cents per pound, and 300 lbs. of mercury are added after the pan has run one hour and is hot. The labour expended is two shifts of two amalgamators on each, at $4½ per day wages. The loss in mercury is about 3 lbs. per ton.

"From the pans the pulp is drawn into the twelve settlers, which run four hours, making 40 revolutions per minute. After running one hour, cold water is let run in and overflow, carrying off the tailings, samples of which are taken for assay.

"The amalgamation proceeds very rapidly at first in the pans. About half of all that is obtained is amalgamated in the first hour and a half; at the end of three hours two-thirds is amalgamated, and after six hours 85 per cent. of what the mill is working to is in the form of amalgam. Nothing material is gained by running the pans beyond eight hours. The mill is working up to from 88 per cent. to 92 per cent. of the assay value of the ore, that being counted as the amount chlorinated. The tailings carry from 12 per cent. to 8 per cent.

"The amalgam is strained in canvas bags and sent to the retort, the charge for which is 2,000 lbs, and the time required seven hours and a half. The fuel used is charcoal, about 8 bushels at 20 cents per bushel (12½ lbs.) to melt one bar of about 1,400 ounces.

"There is one retorter and one melter working one shift a day, wages $4.

"The bullion obtained runs about 600 fine and contains no gold.

"The average grade of ore treated is from $100 to $130 per ton, and the amount treated from 50 to 55 tons per day. When the ore was less base the mill treated 65 tons per day. The salaries charged against the mill are those of the mill superintendent and the assayer. The fuel consumed in the mill is about 15 cords per day, including that used in the two roasting furnaces.

"The following table gives the average cost per ton in labour

and material for treating Ontario ore, taking the actual running expenses, estimated from a production of 50 tons per day.

No. of Men.	Occupation.		Per day.	Per ton.	
1	Foreman		$10·00	20	
1	Assayer		6·00	12	
3	Machinists	at $4	12·00	24	
2	Carpenters	,, 4	8·00	16	
2	Blacksmiths	,, 4	8·00	16	
2	Engineers	,, 4	8·00	16	
2	Foremen	,, 3½	7·00	14	
9	Dry floor men	,, 3½	31·50	63	
3	Battery men	,, 4	12·00	24	
6	Roasters men	,, 4	24·00	48	
12	Cooling floor men . .	,, 4	48·00	96	
4	Carmen	,, 4	16·00	32	
4	Amalgamators	,, 4½	18·00	36	
1	Retorter	,, 4	8	16	
1	Melter	,, 4			
4	Labourers	,, 2½	10	20	
4	Watchmen	,, 3	12	24	
2	Ore floor men	,, 3½	7	15	
3	Clerks	,, 4	12	24	
66			$257·50	$5·15	$5·15

Supplies required.		Per day.	Per ton.	
Salt, 10 tons	at $8·00	$80·00	$1·60	
Quicksilver, 175 lbs.	,, 0·50	87·50	1·75	
Wood, 15 cords	,, 4·50	67·50	3·53	
Coal, 12 tons	,, 8.25	99·00		
Castings			1·50	
Oil and Waste			25	
Sundries, chemicals, &c.			50	
Hauling from mine			49	
Charcoal, assaying, and melting			25	
			$9·87	$9·87
			Total	$15·02

"This does not include office expenses, general superintendence, repairs, insurance."

Working Ores at the Silver King Mine in Arizona.— The Silver King is one of the richest mines in the territory, and for years has been a steady producer, paying large dividends to the stockholders. The ores are what are termed "rebellious," and Mr. Aaron* has given the following details as to their manipulation :—

"When the Kiss lixiviation process was first introduced at the Silver King works the ore was of a character to render that process eminently applicable, being of a rather high grade in point of richness, and consisting largely of fahlore, chlorides, bromides, and oxides, in a gangue of quartz and heavy spar. Zinc blende and galena were not present in any considerable quantity at all.

"The roasting furnace used was that known as the "Pacific Chloridizing Furnace," a simple rotating cylinder of boiler iron lined with bricks, 16 ft. in length by some 6 ft. in diameter, and capable of containing a charge of about five tons of ore. Such a furnace is usually provided with but one fireplace; the flames from which pass over the ore within, the smoke and flames escaping by a flue at the opposite end. Unlike the Brückner furnace, which it otherwise resembles, this has no arrangement for moving the ore from the cooler to the hotter end, and *vice versa*, and in a cylinder of such length a considerable difference of temperature is unavoidable.

"To obviate this difficulty and enable the ore to be sufficiently heated throughout and yet not too highly heated in any part, my predecessor introduced an improvement consisting in placing a firebox at each end of the cylinder and firing each alternately at intervals of an hour, either firebox being connected with the flue at pleasure by the raising of a damper.

"This arrangement has the unquestionable advantage of a more equal heating of the ore. It has the disadvantage of wasting a great deal of fuel; also a portion of the ore is lost by being deposited in the fireboxes, through which the dust, impelled by the draught, must pass on its way to the flue. The dust so deposited among the glowing embers of the recently

* Report of the Director of the United States Mint.

abandoned fire forms clinkers and causes rapid deterioration of the grates. Other minor defects of the system may be passed over.

"It was found that the ore sustained a serious loss of silver by volatilization during the roasting, to prevent which steam was introduced, as was done by Von Patera, who first made practical application of the solubility of silver chloride in the solutions of the alkaline or alkaline-earthy hyposulphites, in 1858, at Joachimsthal, from a suggestion made by Dr. Percy ten years previously. The effect of steam is to decompose the volatile metal chlorides, by which the volatilization of silver is mainly determined, converting them into oxides with simultaneous formation of hydrochloric acid. The latter then assists in the decomposition of remaining sulphurets, and the formation of silver chloride, which of itself is not very volatile, and, though capable of decomposition by steam at an elevated temperature, is not easily decomposed in this way, especially in the presence of salt, chlorine, and hydrochloric acid constantly tending to its production. At this period the character of the ore was such that a charge of 5 tons could be well roasted in from fourteen to sixteen hours, converting about 95 per cent. of the contained silver into chloride.

"The results were so satisfactory that two more furnaces of the same kind were erected, and a respectable output of bullion was obtained.

"Unfortunately, as depth was reached in the mine, the character of the ore changed. The proportion of chloride and fahlore diminished, and zinc-blende and galena became more and more abundant. The roasting became slow and tedious. The percentage of soluble silver in the roasted ore decreased somewhat, causing richer tailings, and as the richness of ore also decreased a serious diminution of the monthly output of bullion was inevitable. The ore also developed a tendency to cling to the lining of the furnace, forming a thick crust on the bricks, necessitating frequent stoppages for the purpose of its removal, and, in the interim, greatly reducing the capacity of the furnaces.

"In the month of April of the current year, 1882, the length of time required for the roasting of an average charge of $3\frac{1}{2}$ tons reached a maximum of thirty-two hours, a figure above 29 appearing on the record for the first time in that month (with the exception of a single instance in March, when the charge is reported as 'spoiled in the furnace').

"When the writer took charge of the works in the month of May, the roasting was continued in the same manner, by the same workmen, and with similar results to those of March and April. The ore, however, was of a somewhat lower grade. In June the average time required to roast a charge increased, and in July the ore became so bad that the men who had been roasting for two years found it impossible to avoid 'balling.'

"The balls formed were peculiar, being perfectly spherical, resembling shot, and of all sizes, from that of a pin's head to that of a marble, extremely hard, and consisting of concentric layers. When a charge is "pilled" in this manner it requires from fifty to seventy hours to "sweeten," and as these pills refuse to soften in the leaching, the extraction of a high percentage of the silver becomes impossible, unless the ore be re-crushed; moreover, the roasting under these conditions is necessarily imperfect.

"Determined to ascertain if it were possible to roast such ore in such furnaces in the ordinary manner, with the addition of 10 per cent. of salt, the writer in person made the attempt, and succeeded in turning out a charge containing but very few balls, and chloridized to 90 per cent. But the care and skill required to effect such a result were too much for the workmen, who continued balling the ore."

I will here call attention to a circumstance which operates very disadvantageously in these works. The ore for the roasting is crushed wet and received in pits dug in the ground. It is thus impossible to prevent the separation of the finer and lighter portions from the coarser and heavier, and although in removing the ore from the pit and drying it in a reverberatory furnace an effort is made to obtain an equal mixture, it unavoidably happens that some of the roasting charges contain

ores without their due proportion of heavy minerals, owing to the imperfect system of concentration going on during the settling in the vats. Thus not only does the ore, as it comes from the mine from day to day, exhibit considerable variations in quality, but these variations are much aggravated by the system of wet crushing and the lack of proper facilities for re-pulverizing, storing, and mixing.

Each furnace charge is sifted by hand-work into an elevator, and the lumps formed in the drying are then pulverized in a Dodge machine, and also elevated. The entire arrangement of this department of the works is faulty in conception and wasteful of ore and labour in operation.

In the month of July, when the ore was not by any means at its worst, analysis showed it to contain 12 per cent. of zinc (equal to about 18 per cent. of blende), 6 per cent. of lead as galena, a good deal of antimony, a little arsenic, and very little iron or copper. It also carried trifling quantities of cadmium, selenium, tellurium, and bismuth. The conjunction of antimonial and plumbiferous minerals with zinc-blende tends to make the roasting difficult.

The character of the gangue exercises a great influence on the roasting. The presence of quartz is advantageous; spar or gypsum is not troublesome, but earthy carbonates are detrimental. Magnesia is bad. In July the ore contained less quartz and spar than formerly, and more of the so-called porphyry, which contains magnesia in abundance.

In the case of ore which balls in the furnace when roasted with salt, the usual practice is to roast without salt to a certain stage, when the salt is added and the heat increased; but the presence of metallic silver and the absence of a fair proportion of iron rendered this method inapplicable. The next idea which suggests itself to the metallurgist is to roast to complete oxidation without salt, and then chloridize by an addition of calcined copperas and salt. This method, with a slight modification, has been used on some of the worst of the ore with very good results.

Another plan, which has been used with success, is to mix a

certain proportion of sand with the ore. A charge of 3 tons of ore with the addition of 7 per cent. of sand, is roasted better and in a shorter time than the same quantity of ore without sand. The sand used contains a little silver, being the coarser portion of a pile of rather rich tailings from former concentrations. It is not always necessary to resort to these expedients, for, as already said, the ore varies, and some pits admit of roasting in the same way as formerly. The addition of one-third of clean quartzose silver ore was found to act favourably, 95 per cent. of chloride being got in twenty-four hours with 3-ton charges; but such ore is not readily procurable.

CHAPTER IX.

ROASTING FURNACES AND KINDRED APPLIANCES.

THE STETEFELDT FURNACE—Automatic Feeder—Brückner's Roasting Cylinder—The Improved Brückner Cylinders, as described by Mr. Raymond—The Advantages Claimed—The Improved White Furnace—The O'Harra Roasting and Chloridizing Furnace—Revolving Ore Dryer—Krom's Dry Kiln.

EXPERIENCE has shown that when ores contain a very large amount of sulphur, mechanical roasting furnaces cannot, as a rule, compete with the old-fashioned reverberatory furnace, with long hearths, such as I described in a previous work.* But with ores which contain a reasonable amount of sulphur, the Stetefeldt and some others of the mechanical roasters are found to give excellent results. In the previous chapter I have referred to the use of the first-named of these furnaces in the milling operations conducted by me at Mineral Hill, and in the following account I draw upon my own experiences of the furnaces at those mills.

The Stetefeldt Roasting Furnace.—The principle of Mr. Stetefeldt's invention consists in the ore falling through an ascending current of flame, whereby the ore is not only oxidized, but when mixed with salt is also chloridized. To Mr. Stetefeldt credit is due for the discovery that a complete chloridizing roasting of silver ores can be effected within the few seconds it takes for a shower of ore and salt to pass through an ascending flame; and when I was apprised of the fact some twenty years ago it was with some incredulity that I

* "The Metallurgy of Gold."

read the report, nor did I feel convinced until after investigation of the process. Other furnaces constructed on the same principle are only an application of this original device.

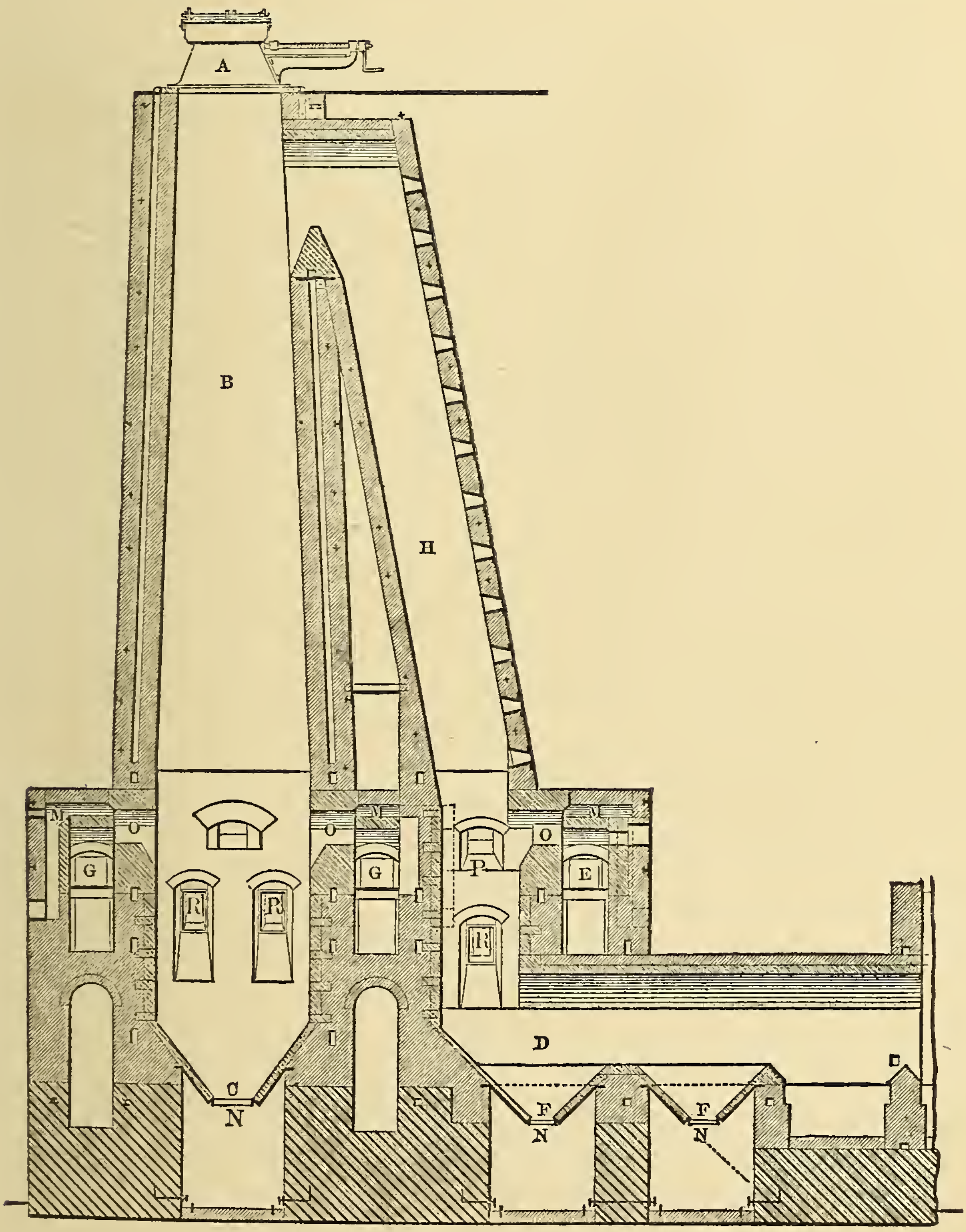


FIG. 59.—STETEFELDT ROASTING FURNACE.

It may be considered, in truth, as one of the best inventions in silver metallurgy, for without its introduction it is doubtful if so many mines in Nevada and elsewhere on the Pacific

coast carrying base metal ores would have been successful, and have led the way (as they have done) to further discoveries and the opening up of new districts.

The furnace (which is shown in Fig. 59) consists of a perpendicular roasting shaft, B, from 26 to 36 ft. in height, heated by the fireplaces, G. The flue, H, is also heated by a fireplace, E, and carries off the waste gases and the dust. When the ore arrives at the hopper placed above the furnace, it is mechanically discharged into the automatic charging apparatus, Fig. 60, which consists of an iron hopper, A, placed on the top of the furnace, provided with a draw valve, B, which is always open when the furnace is in operation. Above this is another cone, on the top of which is a cast-iron grate, C, and on the top of this is a screen made of steel plate punched with holes. Above the screen is a wrought-iron frame, E, on the bottom of which a coarse screen, F, of heavy wire is placed. This frame with its screen rests on friction rollers, G, rotating on brackets, H, which can be raised or lowered by means of set screws so as to have any desired distance beween the punched screen, D, and the wire one, F. The bracket, K, carries an eccentric shaft which is connected with the shaft, M, which moves the frame, E; but as the oscillating motion would not always be sufficient to force the ore through the two screens, stationary blades, O, are fastened to the brackets, N, which can be raised or lowered by the nuts, P, so as to bring them into more or less close contact with the screen, F. The blades distribute the pulp over the screens evenly. The frame, E, is kept in motion by a cone pulley, and is so arranged that it can be made to take any motion that it is desirable to give it. The usual velocity is between twenty and sixty strokes per minute. By changing the distance between the screens, and also the velocity of the movement, any desired amount of ore can be delivered in the furnace with the greatest regularity.

The main shaft, B, was 26 ft. high at Mineral Hill, and was heated by the ascending gases from two gas-generating furnaces, G. Where ordinary furnaces are employed, as shown in Fig. 59, the gases and flame enter the shaft by means of the

opening, O, which communicates with the outside air by means of channels having openings, M, one on a level with O, the other below, and they are closed by an iron sliding door, so as to regulate the quantity of air to be admitted to furnish the required oxygen for the perfect combustion of the gases. The fireplaces are provided with doors at the ash-pits. On a level with the opening, O, in the face of the furnace where the discharge is located, is a door, P, and also one in the return flue, H, also marked P, which are for the purpose of observing the flame and heat in the furnace shaft and return flue, and there

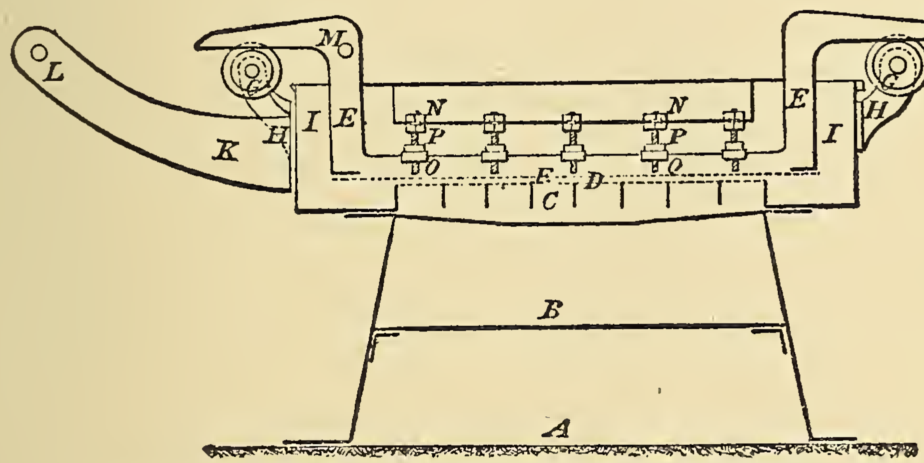


FIG. 60—THE STETEFELDT FURNACE. AUTOMATIC FEEDER.

are also the doors, R, which serve for the introduction of the tools for scraping the walls when any ore sticks to it.

The main shaft is built slightly tapering, and if ores are very rebellious it may be increased in height to even 46 ft. The horizontal section is from 4 to 6 ft. square, giving a surface of 16 to 36 square ft.; and, as is shown in the drawing, the walls of the main stack are double, having an air space between, which keeps the heat regular. Our furnaces being heated by gas generators, all the portions of the furnace exposed to the direct action of the flame and the generators were constructed of the best firebricks. The fuel employed was wood and charcoal mixed, at the beginning of the firing, and then charcoal alone.

One man attended to the firing, one man on top of the stack attended to the automatic feed, and two men to the dis-

charge, all working twelve-hour shifts. The discharge in our furnaces was dissimilar from the one shown in Fig. 54; we had a door at the bottom of the stack, and when a certain amount of ore had accumulated the door was opened and the red-hot pulp drawn with long iron rakes on to the cooling-place in front of the furnace. This was immediately cooled by means of a jet of cold water through a sprinkler; but in the more modern furnaces the ore falls through the shaft into the hopper, C, below, which is closed by a sliding door, N, and from this is discharged into cars on a tramway to the cooling floor, where it is left red-hot for several hours. Experience shows that by leaving the ore in this state the percentage of chlorination is increased 10 to 20 per cent.; but as my chlorination tests always showed 88 to 92 per cent. I hardly think that I should have gained very much by it. Besides, there was always an interval of two to three hours before I discharged the furnace, and the roasted pulp lying at the bottom of the stack in a red-hot state underwent the same action as if it had been on a cooling floor, as air was admitted to the mass through draught holes. Particular attention has to be paid to not having too high a heat, so as to melt or cinder the ore, and also not to have too low a heat. Ores containing lead require constant attention on the part of the firemen, and a lower heat.

As the ascending flame heats the feeding apparatus, shown in Fig. 55, the portion A, B is surrounded by a water jacket, supplied by a small stream of fresh water which enters through a half-inch iron pipe at the bottom of the jacket and escapes at the top.

At Mineral Hill we had a natural water pressure from our supply tank to the top of the stack, but where such pressure cannot be obtained a tank should be provided on top of the building and water pumped into it, and the feeder must watch the gauge to see that the supply never fails. I draw particular attention to this point, as we had a most unfortunate accident happen to us, by which the feeder lost his life. While attending to my duties I heard a loud explosion on top of the building, and rushing up the stairs I found the feed-room filled with

clouds of smoke and dust. Noticing the flames of the uncovered stack, and no answer being given to my questions by the feeder, I hastened back with the alarmed workmen, and on opening the discharge door we were horrified to find the body of the unfortunate man roasting in the red-hot mass of ore. On dragging him out, a seething mass of humanity, we extinguished the fire in the furnace and stopped work. My investigation revealed the following facts: the feeder had left the feed-room, and must have shut off the water supplying the jacket, which was decidedly against the rules; and he was noticed by one man returning to the room, and was only in there a few seconds when the explosion took place. The cast-iron feeding apparatus, although weighing several hundredweight, was lifted bodily from its position and shifted to the side opposite the entrance door. The apparatus was very hot when I examined it—which could not have been the case had the water circulated through it—and the broken supply pipe was discharging a stream of water. I concluded that the unfortunate man had turned off the water; during his absence the water in the jacket partly evaporated and the exposed iron became red-hot, and when on his return he turned on the cold stream an explosion took place, and becoming bewildered by the smoke and dust, he stepped into the furnace shaft, which opened before him like a fiery abyss. I relate this sad experience as a warning to those who may have charge of similar works.

The ascending gases carry a certain proportion of the finest dust over the bridge into the return flue, H, and as these particles are not perfectly chloridized, an additional fireplace is provided at E, and the downward draught carries the flame past P and R into the dust chambers; there are also air holes at M and O. The ore collects in the dust chambers, D, and settles in the hoppers, F, from where it is discharged in the same manner as from C. At Mineral Hill the accumulations from the dust chambers could only be removed once a month, when the mill was closed down for a general clean-up, and the men had to enter the chambers and shovel out the stuff which had accumulated on the floors.

There are a number of openings at s s, to clean out the return flue, as some ores clinker and adhere to the walls. As I never worked my ores at a very high heat and a strong draught, I should judge that about 65 per cent. collected at the bottom of the main shaft, 25 per cent. fell to the bottom of the return flue, and the balance collected in the dust chambers, which were very spacious. The loss of silver through volatilization or otherwise—by many supposed to be considerable—I could never ascertain; and my experience leads me to suppose that no silver was lost during the roasting, or went "up the chimney," which is a popular fallacy. With ores containing large percentages of zinc, antimony, and very volatile products there might be some loss, but with spacious dust chambers this loss can be minimised. The chimney is from 60 ft. to 100 ft. high, and has a horizontal section of from 4 ft. to 5 ft. There is a damper at the end of the dust chambers at the point where the same connects with the mill, and the draught required to work the Stetefeldt furnace must be closely studied; indeed, it requires constant watching.

In the 15-stamp mill we crushed 18 tons of ore in twenty-four hours, but the quantity of ore which can be treated in a Stetefeldt furnace is much larger, and varies, according to the size of furnace and character of the ore, from 30 to 60 tons. If the ores contain much sulphur, like pyrites, galena, zinc-blende, which have to be fed slow, the output is smaller. Such ores also require a larger percentage of salt, and extensive experiments have to be made at each mine to determine what the quantity of salt should be with the different characters of ore.

I also ascertained that the ore in the return flue gave a higher chlorination test than the ore in the main stack. At the general clean-up, when the accumulations of the dust chambers were cleaned out, these were piled up and worked in gradually with the freshly roasted ores.

Cost of Roasting.—The expenses of roasting 18 tons of ore in the Stetefeldt furnace were :—

2 feeders	at $4	=	$8
2 firemen	" 4	=	8
4 dischargers	" 4	=	16
1800 lbs. salt	" 0·06	=	108
Wood and charcoal		=	22
			$162

or $9·00 per ton, equal to £1 16s.

The excessive cost of salt, high price of labour and fuel, account for this abnormal charge, but there are many establishments in existence now which do the roasting for 16s. to £1 per ton in the Stetefeldt furnace.

Specifications of Stetefeldt Furnaces.—The following is a list of items of material used in the construction of a Stetefeldt furnace:—

	No. 1 Ex.	No. 1.	No. 2.	No. 3.
	lbs.	lbs.	lbs.	lbs.
Castings	17,300	15,800	13,600	9,600
Wrought Iron	5,000	4,800	3,350	2,500
Bolts	4,000	4,000	2,800	2,350
Buckstays	10,650	10,550	6,150	4,980
Bars	8,850	8,950	4,650	3,850
Sundries	2,500	2,500	2,500	2,500
	500	500	500	500
Totals	48,800	47,100	32,950	26,250

The weight of two dozen punched screens and 10 ft. of wire cloth is about 650 lbs. additional to above totals.

	No. 1 Ex	No. 1.	No. 2.	No. 3.
	cubic feet.	cubic feet.	cubic feet.	cubic feet.
Stone	3,000	3,000	2,500	2,000
Brick	260,000	260,000	210,000	150,000
Fire-brick	4,500	4,000	3,500	2,500

The above for construction of furnaces, dust chambers, and 100 ft. of flue between dust chamber and chimney.

The capacity of No. 1 Extra and No. 1, 30 tons to 60 tons in 24 hours; No. 2, 20 to 40 tons; and No. 3, 10 to 20 tons. For very base ores the furnaces will work respectively 30, 20, and 10 tons; and for ordinary ores, 50, 30, and 15 tons. No. 1 Extra is only for very base ores.

W. Bruckner's Roasting Cylinder has been in successful operation in America and Europe. Lately considerable improvements have been made, increasing capacity and securing good results for roasting ores for amalgamation, lixiviation, and smelting. This furnace may also be used for the manufacture of sulphuric acid out of pyrites or other kinds of sulphurets.

In the accompanying drawings we have in Fig. 61 sectional views through the apparatus, and in Figs. 62 and 63 sectional views of a modified arrangement. Like letters refer to like parts in each view.

A represents a furnace provided with suitable feed and cleaning doors, and with a grate, B. Furnace A communicates through a pipe, C, with a chamber, D, which (as shown in Fig. 61) is supported upon suitable pillars, E. Chamber D is formed preferably of an iron shell lined with brick, and its bottom plate is made of cast iron, there being formed thereon a wall of masonry, F, which is inclined toward a pipe, G, which connects said chamber with the revolving cylinder, H, as will be described. Tapped into pipe C is a blast nozzle, I, and into pipe G a nozzle, K. The operation of the furnace is described by the inventor as follows:—

"The fire is built upon grates B, and the gases arising therefrom pass through pipe C to chamber D. Ore, in a powdered form, is then fed through nozzle I, and forced upward in said chamber, and, commingling with said gases, the sulphur therein contained is ignited. The ore thus heated then falls in the form of numberless sparks, either directly into pipe G or upon incline F (by which it is fed to said pipe); and upon thus entering the pipe G it is forced by the blast from nozzle K into the revolving cylinder, H. Cylinder H has end bearings in wall

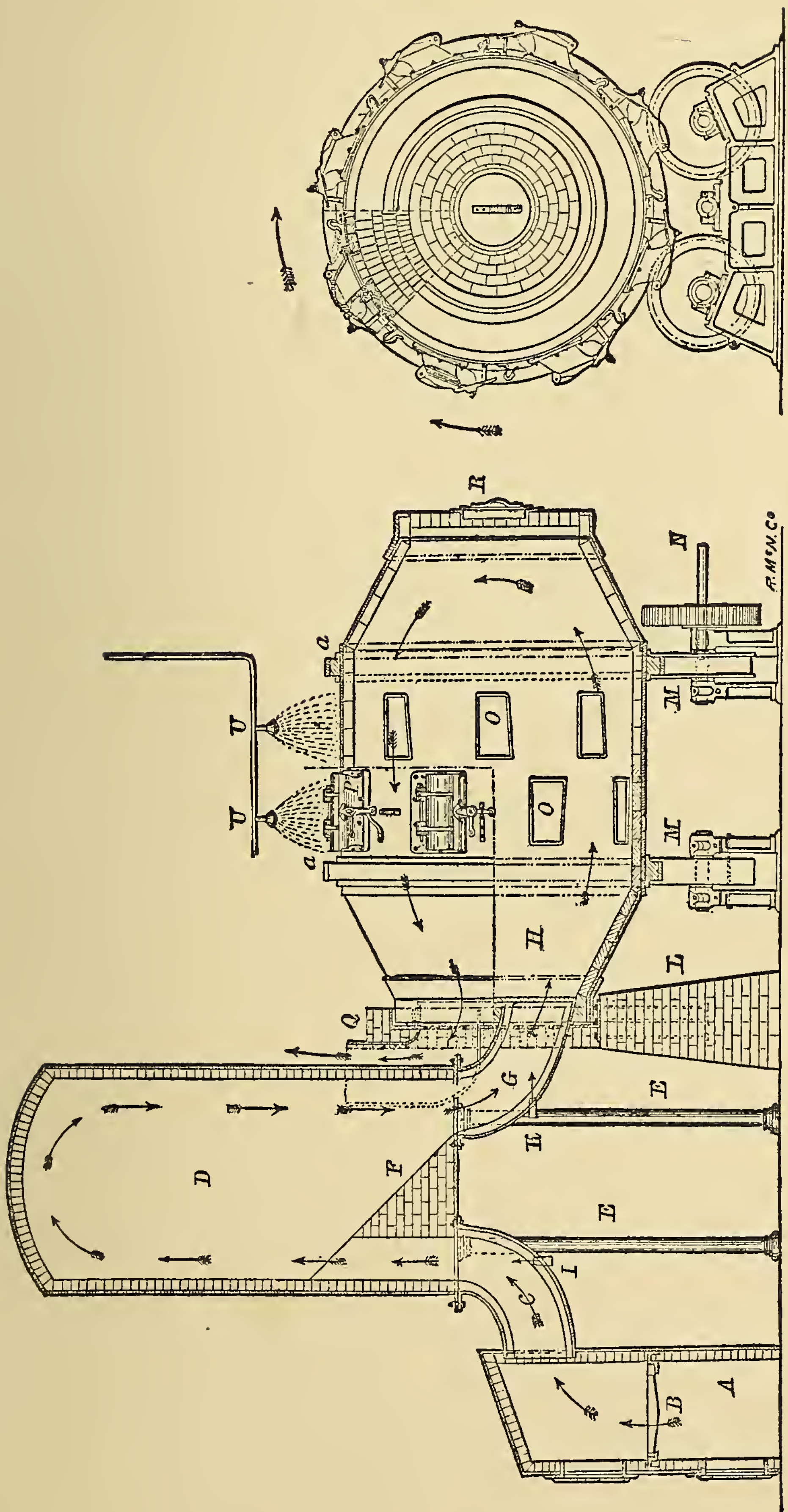


FIG. 61.—BRUCKNER'S ROASTING CYLINDER.

L, and is provided upon its outer face with one or more flanges, *a*, which ride upon grooved or plain rollers, M. Two of said rollers are driven by suitable power from the shaft N, while the others are driven through friction; or, if desired, power may be imparted to the cylinder in any other well-known way. Cylinder H consists of, preferably, a metal shell lined with brick, and has formed through it two or more series of openings, surrounded by frames secured upon the outer face of the cylinder, and arranged to form pockets, O. Each pocket

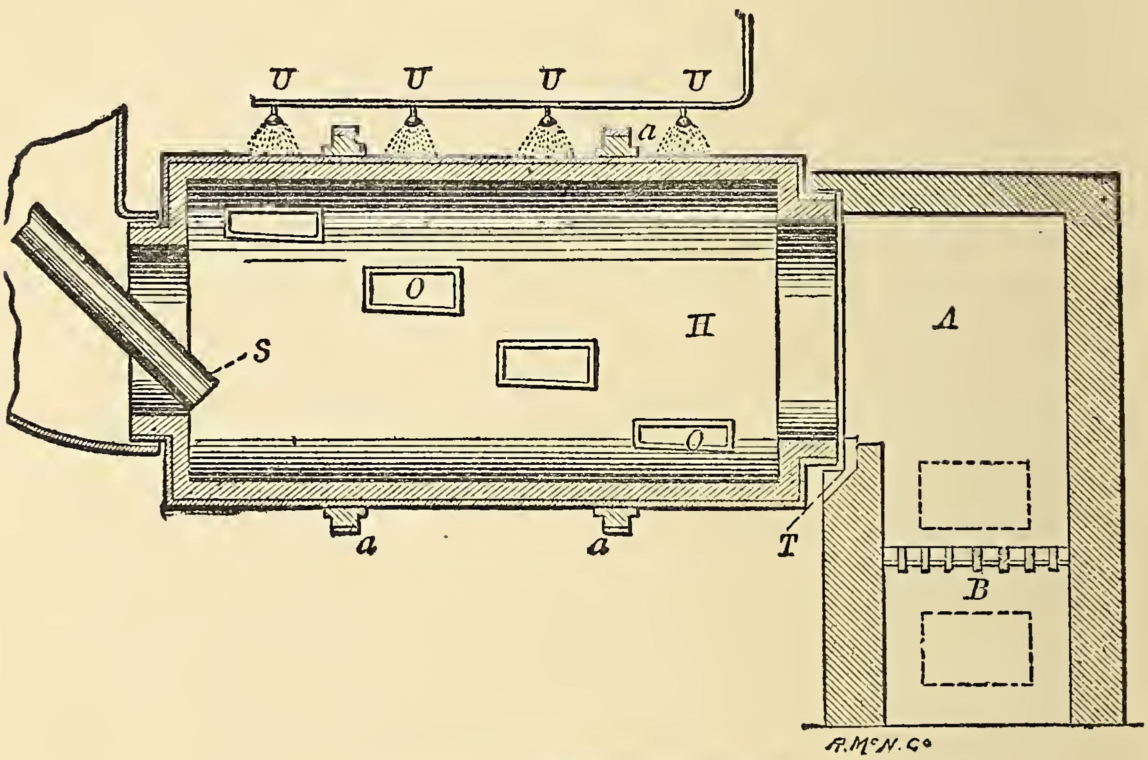


FIG. 62.—BRUCKNER FURNACE.

thus formed is closed by a hinged door, P, which may be opened when desired, but which otherwise is held closed by a lever or catch, *h*. All the buckets of one series, nearest the feeding end of the cylinder, are formed at an incline to the cylinder, in order that the ore elevated thereby may be thrown toward the opposite end of the cylinder; while those of the other series are placed at an opposite incline, to throw the ore back to be elevated by the first series. It will be understood that upon the revolution of the cylinder the ore fed thereto, in settling in the bottom, will fall into one of the first series of pockets,

and by it be elevated until such pocket occupies a position to cause such ore to be discharged therefrom down through the flame or hot-air blast, to be again elevated and discharged by the second series of pockets.

"The blast from pipe G passes through the cylinder as indicated by the arrows in Fig. 61, for finishing the roasting process with or without the use of salt and fuel, and carries off the gases through a discharge pipe, Q, connecting with suitable dust chambers. R represents a door closing the end of the cylinder, through which access may be obtained to the interior thereof for cleaning, repairing, &c. Upon the outer face of the elevator pockets it will be found advantageous to discharge a steam or water spray from a sprinkler, U, for the purpose of cooling the same to avoid wear. In Fig. 62 is shown a modification of the apparatus herein before described. In this figure the elevating pockets with which the cylinder is provided are arranged in an oblique line with respect to one another, and are all inclined in the same direction, so that ore fed through feed-pipes will be taken up by said pockets and gradually carried to the discharge opening, T.

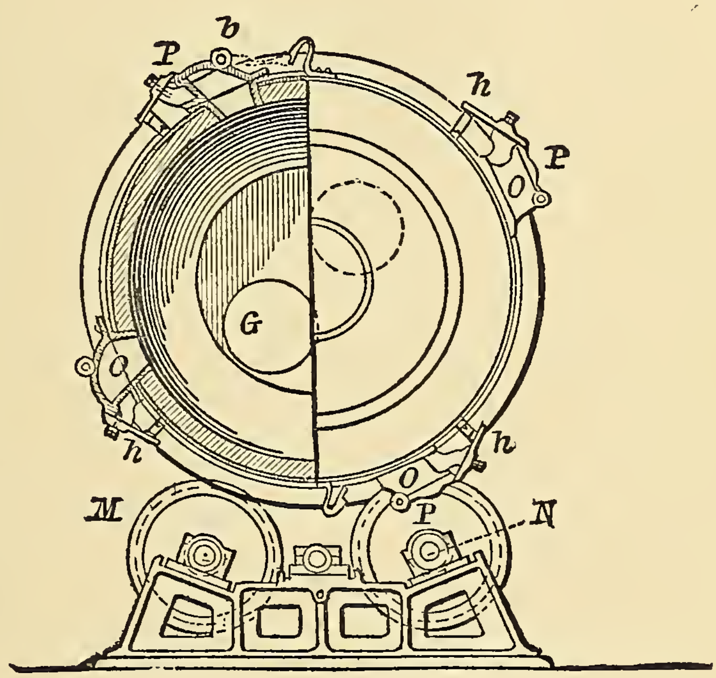


Fig. 63.—End Elevation of Bruckner Furnace.

"By providing the cylinder with the pockets described, more ready access to the interior is obtained for the removal of any ore that may stick to its inner face, with, at the same time, ready means for the discharge of the roasted ore; and by the arrangement of said buckets and discharge of water

or steam upon their outer surface, the injurious effect of heated gases thereon is avoided."*

Although the cylinder can be lined with fire-brick, ordinary bricks laid in a mortar of fire-clay and sand will last a long time. Dust chambers are connected with the flue, which are cleaned out occasionally, and it is advisable to introduce a jet of steam into these chambers, with the object of moistening the fine dust and causing it to fall.

Ores containing large quantities of iron pyrites and galena are difficult to roast in these cylinders, as they clinker, and are not affected by the salt. In such cases the ore ought to be roasted very slowly, and the quantity introduced into the furnace should be also smaller, not exceeding 1½ tons, and the time required for roasting may vary from six to eight hours. If ores contain only a small percentage of sulphides, 5,000 lbs. to 6,000 lbs. can be introduced into the furnace, and the roasting finished in about four hours. The ore is charged by means of a hopper through a circular opening into some cylinders, or by a jet, as shown in Fig. 61, and then set revolving, making one or two turns a minute. With ores which do not contain much sulphur the salt is introduced with the ore; but with heavy sulphuret ores it requires several hours before nearly all the sulphur has burnt away, and during that time a low fire should be maintained, and then the salt should be added if a chloridizing roasting is needed, and the heat raised to cherry red, and the cylinder revolved for five or six hours more. As with the Stetefeldt furnace, regular chlorination tests should be made with hyposulphite of soda. When the roasting is completed the door is opened and the cylinder allowed to revolve, and the roasted pulp drops into an iron ore waggon underneath and carried to the cooling floor. The pulp should emit the peculiar pungent smell of chlorine. The ore ought not to contain lumps or have sintered; these have to be screened out, and crushed again with the scrapings from the cylinders and re-roasted. The quantity of fuel required per ton of ore varies

* In my "Metallurgy of Gold" will be found a description of a different model of the Brückner furnace.

according to the character of the ore, and runs from three-quarters of a cord to a cord of wood, or from half a ton to three-quarters of a ton of coal, per ton of ore.

It is very difficult to say how many tons of ore a Brückner furnace will treat in twenty-four hours, as in some cases it may only be 3 tons, or 4 tons, and yet may go up to 10 tons, and the cost of roasting will vary from 10s. to £1.

The chloridizing is done fully as well in this furnace as in the Stetefeldt, and for small mills it is very advantageous.

Brückner's Two-Cylinder Roaster.* — An improved form of roaster, which has been proposed by Mr. Brückner, involves the construction of two cylinders instead of one, and he thinks that three cylinders might be better still, as securing an exact mechanical reproduction of the work of the well-known three-hearth Fortschaufelungsofen—the most perfect of the reverberatories—in which delicate roasting is performed with hand labour.

The cylinders may be put up at any locality, or added to existing works, but it is preferable to choose a site on a hill sloping naturally, say about 30° or 33°. Two excavations are made, one 8 to 12 ft. higher up than the other, and each is protected by a wall with a batter of about 1 in. to the foot to prevent slides of stone and earth. Each cylinder being 20 ft. long and 7 ft. in diameter, the total excavation should be 60 to 70 ft. along the hill, and 15 to 25 ft. wide. On each level a building is erected 35 ft. long by 20 ft. wide and 12 ft. high, with the roof containing a ventilator against the hill. When both buildings are on the same level, there should preferably be a solid framework of very heavy timber to carry the four rollers which bear the upper cylinder. The four rollers carrying the lower cylinder should preferably rest on two heavy pieces of timber, strongly braced together and laid to their top in a foundation made of stone or brick and cement. Foundations are necessary only under each two opposite rollers, but they must be

* As described by Mr. R. W. Raymond, in a paper read before the Institute of AmericanMining Engineers.

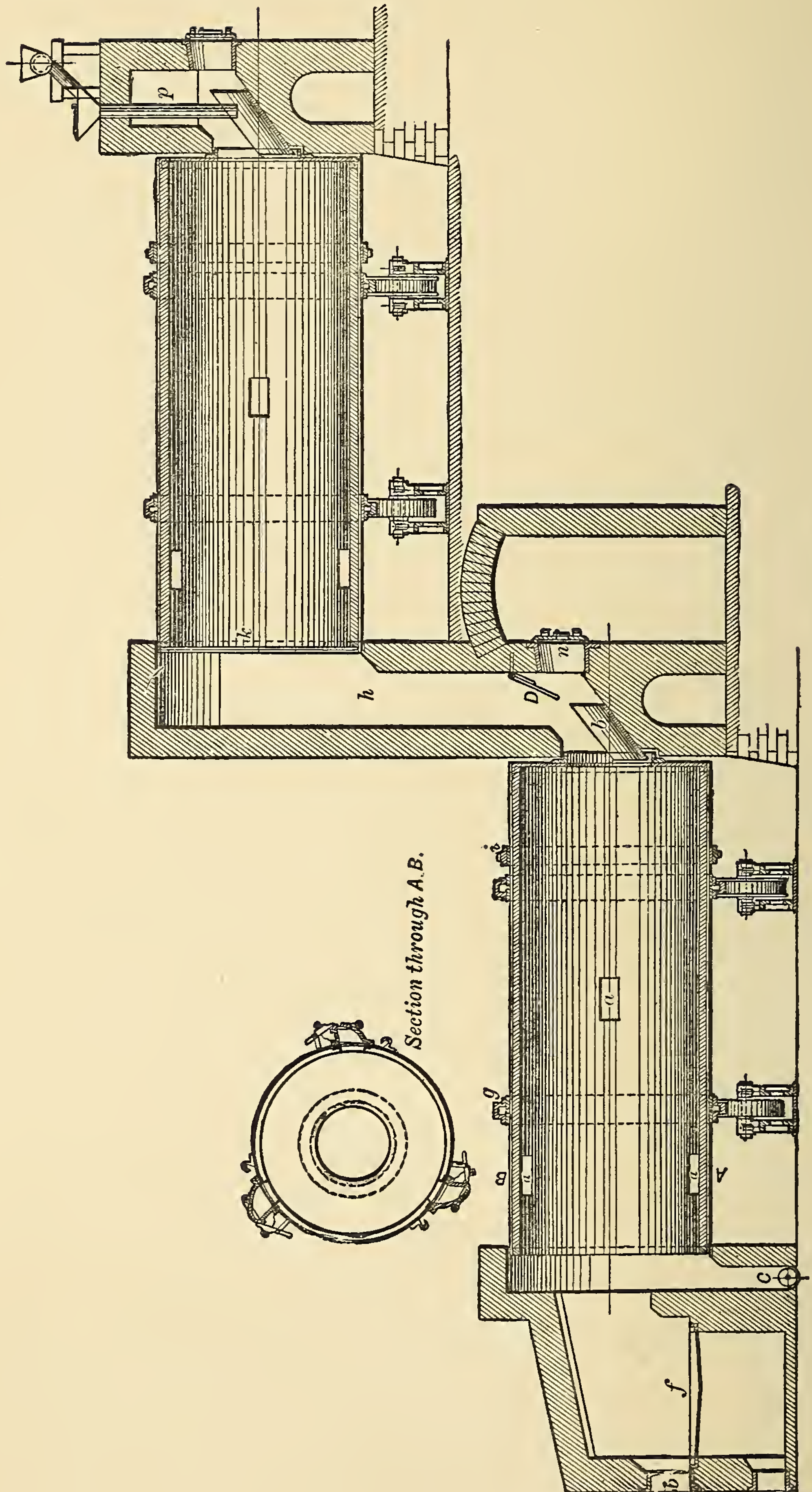


Fig. 64.—Bruckner Two-Cylinder Roaster.

made carefully, because some little vibration in the movement of the cylinder cannot be avoided.

Lining of the Cylinders.—Many persons believe that it is necessary to have fire-brick made in circular segments of the diameter of the cylinder. This is a good thing, but unnecessarily expensive. Common brick, laid flat (not on edge) into good fire-clay mortar, will wear for many years without loosening or abrasion, being protected in most cases by a crust of sintered ore which forms on them. To make the mortar, Mr. Brückner uses two parts of ground fire-brick to one part of ground fire-clay, and a small percentage of salt, which aids in cementing the brick to the outer shell. No bracing is required; the lining forms a ring about 3 in. in thickness, which is, of course, a double arch, and braces itself. The 3 in. of the thickness of the lining furnishes a sufficiently bad conductor of heat, so that when it is red-hot inside the outside shell is hardly ever hotter than 100° C. The expansion is about the same in the inside brick and the outside iron. This thickness of lining is recommended for the iron stack also.

It is a common mistake to think that iron is rapidly destroyed in the presence of sulphur or sulphurous acid. Everybody who has ever made sulphide of iron for the manufacture of sulphuretted hydrogen knows that he has to heat the iron first to *white* heat before he adds the sulphur, otherwise it would not smelt and run out at all. At lower heat the action is very slight and slow. An iron stack, even without lining, lasts for many years *if always in operation.* Of course, if the furnace is idle half the time it corrodes by exposure to air and moisture. It is therefore best, if a substantial and cheap stack is required, to make it 3 or 4 ft. in diameter of No. 10 iron, and line it with brick in the manner explained above. A height of 60 ft. is considered sufficient; a damper ought to be provided at the bottom flue leading to the stack, for the purpose of regulating draught.

Fireplaces.—For the purpose of lightness and convenience of easy erection, Mr. Brückner makes his fire-box of iron lined with 5 in. of fire-brick. For wood, the grate is $5\frac{1}{2}$ ft. long by

30 in. wide; for coal, square, with about the same area. These

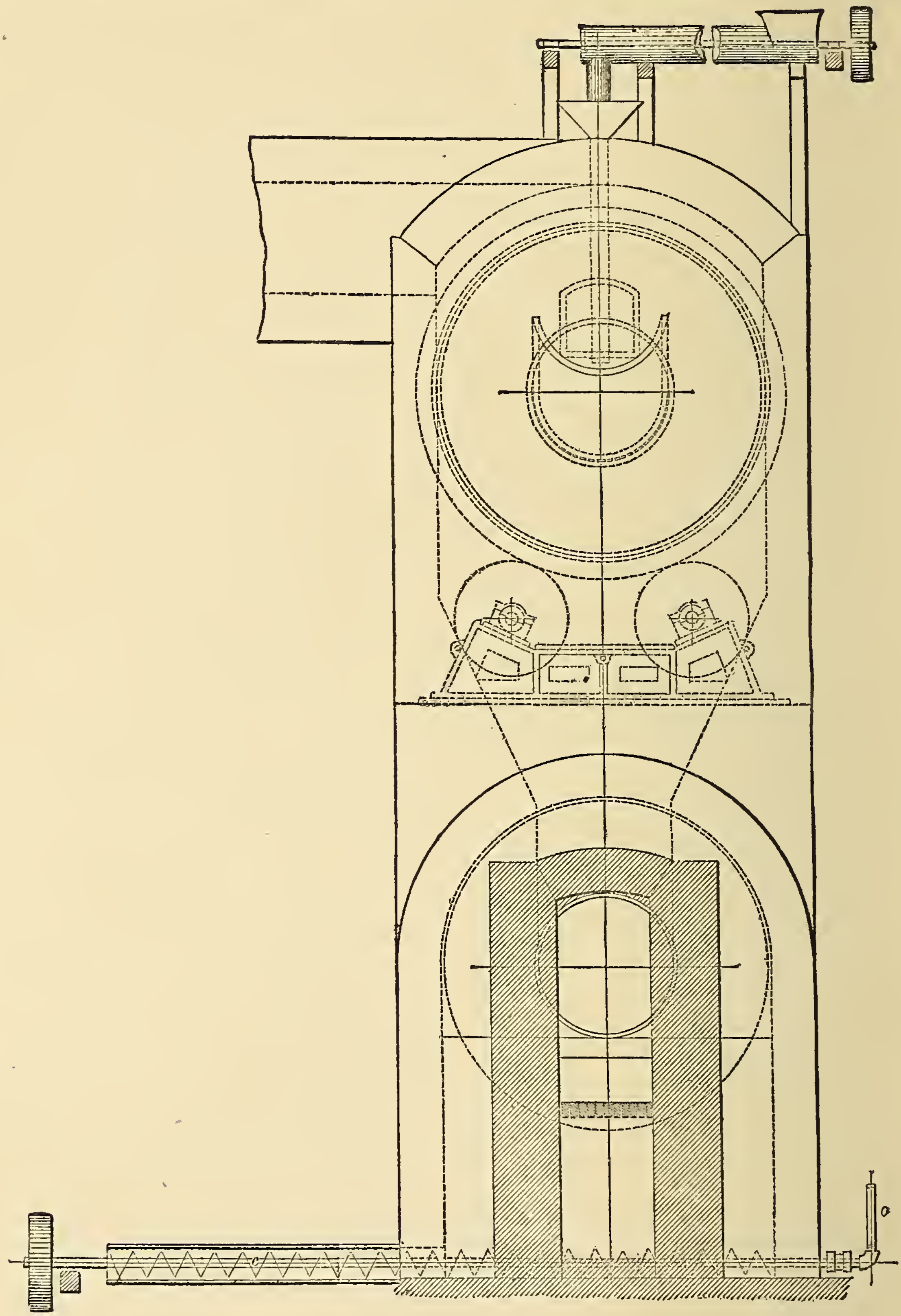


FIG. 65.—BRUCKNER FURNACE.

fire-boxes can be set on small rollers, so as to be easily re-

moved when the cylinder is to be cleansed from incrustations. Fig. 66.

To regulate the heat, it is well to have a second fire-box on one side of the flue connecting the two cylinders, and a third may be useful to roast the raw flue-dust before it enters the condensing chambers.

Dust and Condensing Chambers.—The precious metals are volatilized in roasting, especially in the presence of chlorine, arsenic, antimony, lead, and zinc. Mr. A. Raht, at present superintendent of reduction works in Montana, says that, according to his experience, the German process is not adapted to the ores of that locality when they contain much zinc, arsenic, &c., and that it is preferable to roast such ores at low heat, and not to sinter to a slag, which would cause much loss. Everybody advises low temperature. It is also generally admitted that the volatilization in Stetefeldt's instantaneous roasting process is much less than in that of the reverberatory. The dust and condensing chambers are evidently of great importance. The common construction is that of long flues leading to a chimney. The best results have been obtained in flues by dividing the draft with screens. Filtering through towers filled with wet coke is employed to condense the Cl and HCl gases in the roasting of Spanish pyrites in England, and at Oker, in Germany.

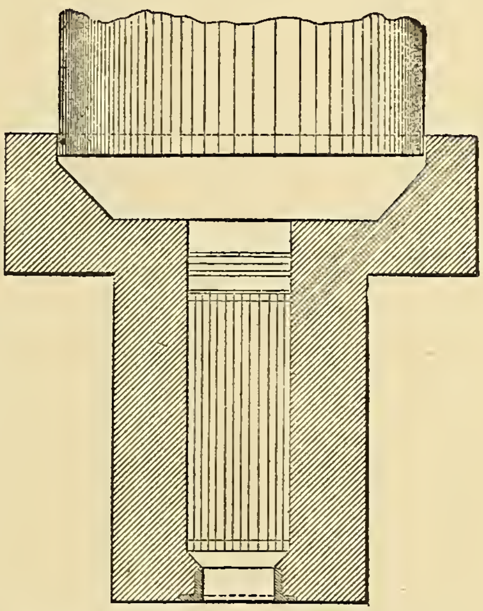


Fig. 66.—Horizontal Section of Fire-place.

Mr. Brückner proposes several towers between flue and stack, filled with balls of glass or other suitable material, resting on a grate and kept clean by a spray of water. The contents of the flue are introduced into these towers by means of a Körting steam-blast, and filtered through the columns of balls. The different towers are used alternately at short intervals,

being so arranged that when one is shut up by a valve, the contents of the flue are forced through the next, and so on. He regards it as not absolutely necessary to have any valves at all. All the towers may have a common flue, and each its own steam-blast. Blast No. 1 drives the contents of the flue into tower No. 1; and when that blast is shut off, and blast No. 2 let on, tower No. 2 will be filled, and so on to the end of the towers, when blast No. 1 commences again.

Mr. Brückner has also devised an ingenious arrangement by which a single tower, 20 ft. in diameter, suffices to do the work of the series just described. The details are withheld because he has not yet made application for a patent.

The Operation: Mixing the Charge.—By far the most important part of the operation is the correct mixing of the charge. In fact, it is almost as essential to success as the right mixing of the charge in smelting. In the latter case the charge is proportioned to get a good slag; in roasting, the ores are mixed to prevent caking, and to get rid of antimony and arsenic.

Moreover, different charges are prepared, according to the subsequent treatment by smelting, lixiviation, or amalgamation which the product is to receive. Much attention was given to this matter at the Halsbrücker works at Freiberg, when amalgamation and lixiviation were used before these processes were more or less superseded by smelting; and the perfection of these celebrated works was due to the large amount of iron pyrites added to the charge before roasting. Mr. Brückner reports that he was indebted in Peru to a very large bank of iron pyrites which cropped out close by his works at Morecocha, and with the aid of which he succeeded in chloridizing the very refractory ores of Perrac, consisting of arsenical pyrites and arsenical galena. Smelting was out of the question, because at that high elevation of the Cordilleras (16,000 ft. above the sea) fuel is very scarce.

Formerly it was thought necessary to have a certain percentage of sulphur in the charge for its effect in chloridizing. This, Mr. Brückner thinks, is a mistake. In his judgment, the iron pyrites is added for volatilizing the arsenic and antimony,

and for preventing the formation of arseniates and antimoniates of silver and gold, which resist chlorination. Very little sulphur or, in its absence, quartz, is sufficient to evolve, when in contact with a very small percentage of salt, the chlorine necessary for chloridizing all the silver contained in the charge after the arsenic and antimony have been driven off.

Thus, an addition of from 1 to 10 per cent. of iron pyrites is in most cases required for the success of the roasting process. Another addition has to be made to such ores as are apt to cake, even at the low temperature of the roasting furnace; it consists of 10 to 25 per cent. of rich tailings. Finally, ores containing lime and alumina have to be mixed with siliceous ores, so that silica shall be in excess.

Mr. Brückner recommends also, in many cases, the addition of a small percentage of copper—pyrites or any other copper—ore, which will be found very beneficial, rendering unnecessary all "doctoring" with bluestone, sulphuric acid, zinc, lime, salt, &c., which are not required when the charge is made right *before* roasting. Chloridizing on the cooling-floor is thereby obviated.

Charging and Discharging.—When concentrations from the jig are to be roasted for the smelting process, the cylinders have to be arranged so as to work continuously. As is shown in the drawing, the discharging end is left open. The ore, wet and coarse as it comes from the jig, is conveyed from the hopper into the end of the first cylinder. There it becomes mixed with a layer of hot ore, dries quickly, becomes ignited, travels down the natural incline through the flue into the second cylinder, proceeds down the natural incline formed by the roasting ore, and is discharged into the cooling-conveyer, which has a hollow shaft, or water jacket, containing flowing water; thence it is conveyed or elevated into the hopper above the smelting furnace.

Mr. Brückner does not recommend this continuous roasting when the subsequent process is to be lixiviation or amalgamation. In such cases there is a much better control if the ore is roasted in charges. For this purpose the lower end is closed

by a flange, and a hopper, large enough to hold a charge of 5 tons, is used above the little charging hopper-pipe, into which it is caused to charge by means of a screw-conveyer, so as to regulate gradual feed. Suppose the last charge has just been let down, through the opening provided for this purpose in the end flange, into the lower cylinder. A small amount of red-hot ore is left in the upper cylinder. Now the new charge is gradually let down from the hopper mixed with the hot oxides left from the former charge, becomes hot, and ignites, until the whole charge is put in. After it has been partly desulphurized, the door at the lower end is opened, and the upper charge drops gradually into the lower cylinder, which has been just discharged into the cooling-conveyer.

After roasting dead, salt or other chemicals are added through the working-door at the back end of the lower cylinder, and the roasting is continued until a sample, taken from the same working-door, shows the desired reaction.

It is scarcely necessary to point out that the ore is by this apparatus charged, roasted, discharged, and automatically conveyed to the hopper above the amalgamator or lixiviating tank, thus doing away with handling the ore, and with the very unhealthy labour of the cooling-floor.

The cost of roasting, per day, is estimated thus :—

Two roasters, at $4·00	$8·00
Two cords of wood, at $5·00	10·00
Oil, light, and extras	2·00
Total	$20·00

Product, 20 to 40 tons of refractory ore roasted in twenty-four hours, the amount being dependent (in inverse proportion) on the proportion of sulphur.

Mr. Brückner claims for his improved apparatus the following advantages :—

1. That for highly sulphurous ores there is a minimum consumption of fuel, the combustion of sulphur in the lower cylinder sufficing to ignite the charge in the upper one.

2. That there is a considerable saving of time as compared with the single cylinder, which requires one to two hours heating to bring a cold charge to the glowing-point.

3. That a better control is possible, since the working-door at the back-end of the lower cylinder permits the easy removal of incrustations, and also the convenient introduction of salt and other chemicals, *after* the partial removal of sulphur in the upper cylinder. This, again, effects a saving of salt, and avoids the formation of a superfluous lot of useless or troublesome soda sulphate.

4. The arrangements for charging and discharging are convenient and save labour, and particularly the water-cooled conveyer takes the place of a cumbrous cooling-floor.

5. Dust and injurious vapours are almost wholly avoided, or rendered harmless.

6. Some further peculiarities in gearing, &c., permitting high speed (which Mr. Brückner favours), are presented as novel and important improvements, but as they are not shown in the drawings, they are not mentioned here.

The following is Mr. Brückner's estimate of cost for a double-cylinder plant:—

Iron-work—2 cylinders, 20 ft. by 7 ft., 2 fire-boxes, 1 smoke-stack, 60 ft. by 30 in., 2 conveyers, 1 hopper and driving gear, driving shafts, countershaft, pulleys, pillow-blocks, &c., complete		£1,170
Freight—40 tons freight to the mine, at 1d. per pound		320
Materials—2,000 common bricks, at £5	£100	
16 barrels fire-clay, lime, and cement	50	
		150
Labour—1 machinist and 2 bricklayers for 20 days	75	
Three helpers for twenty days	40	
		115
Building—60 ft. by 25 ft.		500
Total		£2,255

The Improved White-Roasting Furnace consists of a cast-iron revolving cylinder inclined toward the fire end, and fed at the upper end with crushed pulp. The cylinder is made in sections to facilitate transportation. It is supported on four rings or wheels resting on truck-wheels, and guided in a central position by rollers in upright frames, and revolved by friction of the truck-wheels through gears and pulleys. The angle of inclination is changeable. The cylinder is lined at the mill with fire-brick throughout, and projecting bricks raise portions of the pulp and drop it through the flames, assisting the process. Salt, for chloridizing, is added before the pulp enters the cylinder. The furnace is shown in Fig. 67.

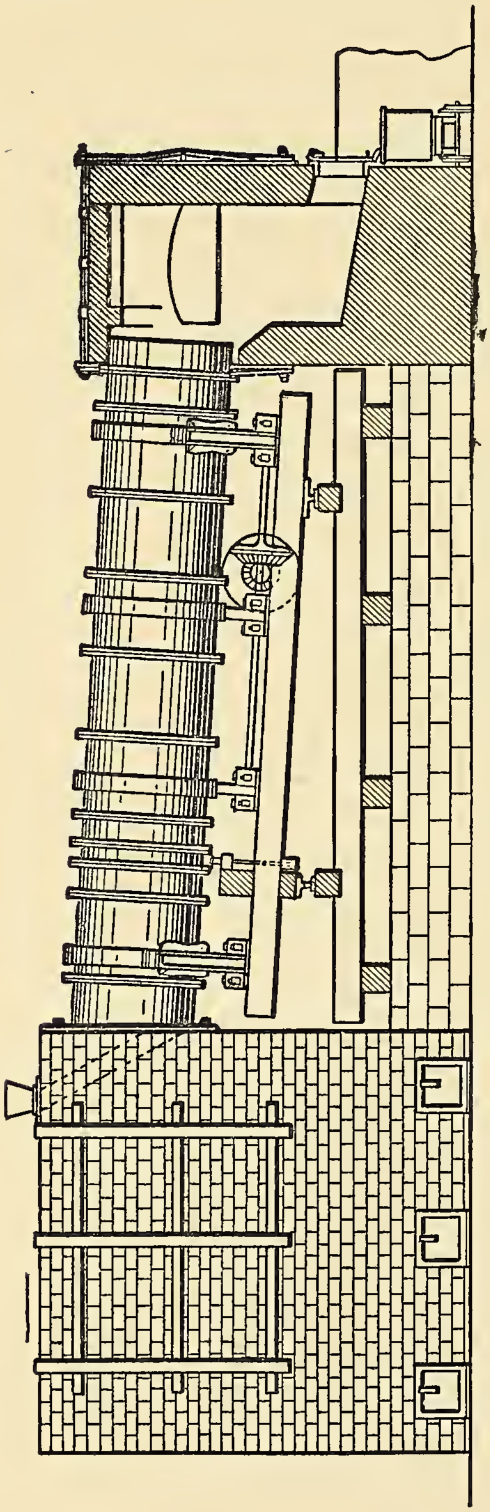


Fig. 67.—The White-Roasting Furnace.

The advantages claimed for this furnace are that it is continuous in its operation, discharging its product regularly into a pit at the lower end, so that the roasted pulp can be withdrawn as required; also that it submits the ore to a gradually increasing temperature, which is the true theory of perfect roasting. By changing the inclination the ore can be retained to a longer or shorter period, as necessary.

The furnace is made in three sizes, as follows:—

40 inches by 24 feet	. .	capacity 15 to 20 tons of ore.
52 „ „ 27 „	. .	„ 20 „ 30 „ „
60 „ „ 27 „	. .	„ 30 „ 45 „ „

The O'Harra Roasting and Chloridizing Furnace.—This furnace is constructed of the usual materials, and is made with two separate hearths, one for desulphurising and the other for chloridizing the ore, both processes being performed at the one operation. It is shown in Fig. 68.

Attached to an endless chain, at proper distances apart, are iron frames formed into a triangular shape; on these frames are a number of ploughs or hoes, set at an angle. One set turns the ore towards the centre, the next set turns it in an opposite direction toward the walls. These ploughs move through the ore every minute and expose a new surface of ore to the flames and gases.

The space between the roof and hearth of each compartment is quite small, so as to confine the heat close to the ore.

The operation of the furnace is as follows: the ore is fed continually from the battery into the hopper, through which it then falls on the upper hearth. The ploughs, actuated by the endless chain, stir the ore over and over on the hearth, and move it gradually to the opening, where it falls to the lower hearth. As the ore is passed along in the upper compartment it is thoroughly desulphurised by the heat furnished by the fire, as described, and by the combustion of the sulphur in the ore. This action is assisted by the oxygen in the supply, as admitted at intervals through the sides of the furnace by the openings. For a chloridizing roasting, salt is mixed with the ore as it is fed into the hopper, and becomes thoroughly intermingled with it by the stirring action of the ploughs. If there is any free silver in the ore it gets the benefit of the chlorine vapours passing up from the lower hearth.

When the ore falls through the opening and on to the lower hearth, the fall breaks any spongy lumps or masses that may have been formed, and the ore is again stirred over and over, and moved along through the flame and gases over the lower

hearth by the action of the ploughs towards the discharging openings.

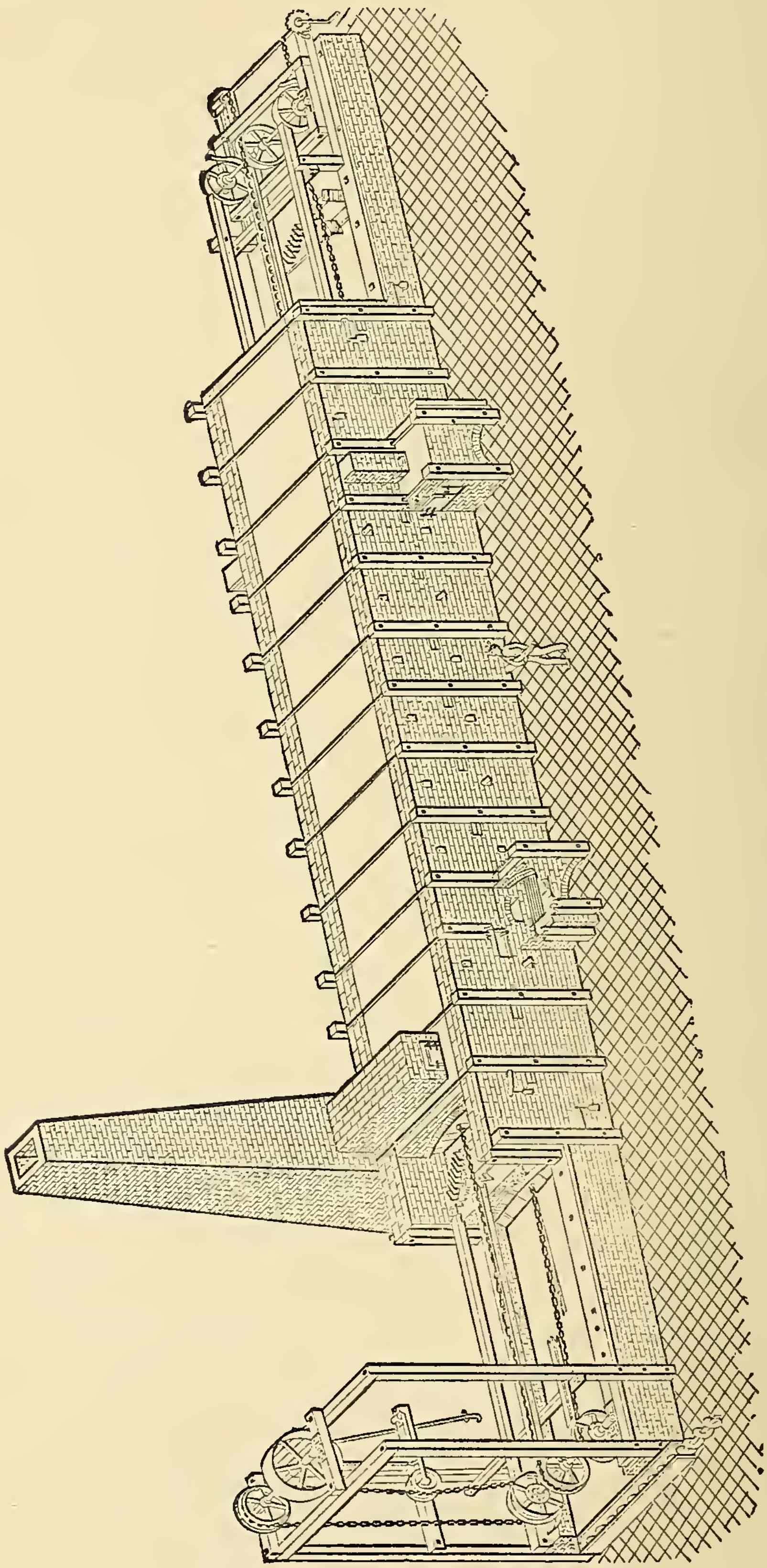


Fig. 68.—O'Harra Roasting Furnace.

The ore has become gradually more and more heated in its

passage through the upper hearth, and by the time the extra heat is required, as stated, it comes immediately in front of the same fires which have during the whole process furnished the heat.

Ordinarily the ore will be from five to ten hours in passing through the furnace, according to its character. Only one man is required to attend the furnace, no other attention being necessary, as the ore may be fed to the furnace by mechanical means, and discharged from the furnace in a car, conveyer, or elevator, and discharged in hoppers over the pans.

The materials required for the O'Harra furnace are :—

Metal.	lbs.
Bolts	1,100
Wrought iron	6,600
Cast iron	3,125
Total	10,825
Brick and Stone.	
Furnace	137,000
Stack, 80 ft. high, 7 ft. by 7 ft. base, 2 ft. by 2 ft. flue, built hollow walls	75,000
Total	212,000

Stones, 125 cub. yds.; fire bricks, 100 cub. yds. or more. Length of hearth, 60 ft.; two cooling hearths (30 ft. each), 60 ft.; total 120 ft. Width of hearth, 8 ft. Width over all, 14 ft. Height over all, 11 ft. Will work up 50 tons. Burns 3 to $3\frac{1}{2}$ cords of wood per day. The top of the furnace may be covered with cast-iron plates, making a first-class dry-floor. It is continuous in its working, and requires but three horse-power for working. A furnace of smaller dimensions, for working 20 tons, will cost, of course, somewhat less.*

Revolving Ore Dryer.—This is an appliance for drying ore as it comes from the mine, preparatory to dry-crushing, and is

* Descriptions of the Bowers and Thompson furnaces will be found in Chapter XV., *post*, pp. 347—354.

shown in the accompanying diagram (Fig. 69). The revolving cylinder, for economy, takes the place of the boiler-iron floor-plates formerly used for ore drying, and under which the flames passed from the furnace. It consists of a cast-iron cylinder in several sections for convenience of handling, having two tracks or tyres on which it rotates, supported by rollers underneath. The motion is transmitted through gearing and pulleys.

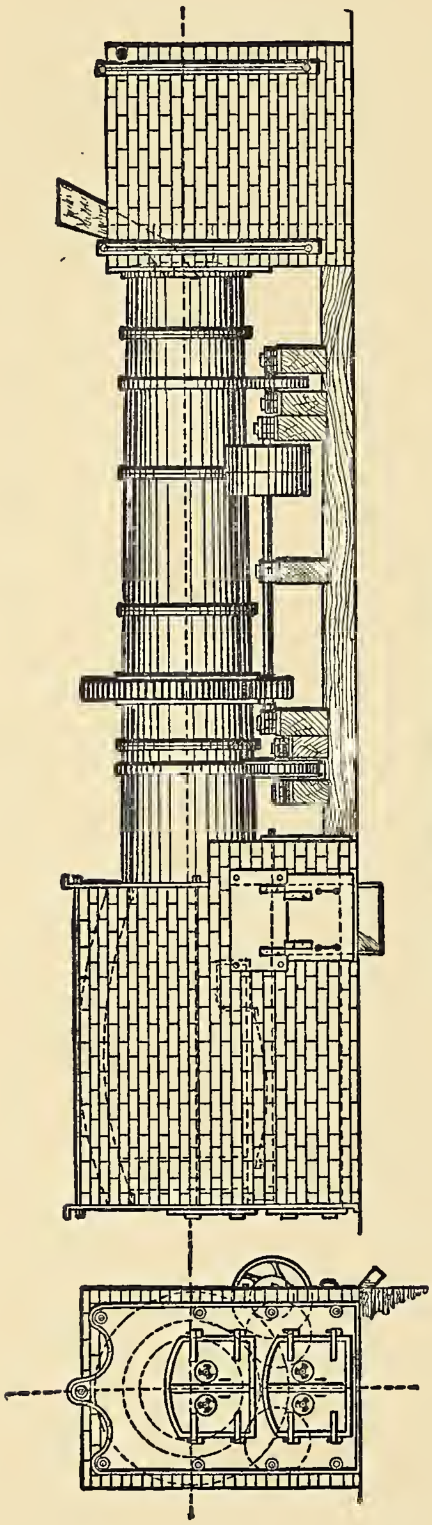


Fig. 69.—Revolving Ore Dryer.

The cylinder is of larger diameter at the fire end, and ore from the rock-breaker is fed at the smaller end. The cylinder's axis is placed horizontally, but owing to its conical form the ore must travel gradually toward the fire at the larger end. Shelves or wings, arranged spirally inside, raise the ore and shower it through the flames, assisting to quickly and thoroughly dry it. The dried ore is dropped immediately into a pit, from which it is drawn through a cast-iron door, and by means of shutes, lined with sheet iron, is conveyed to the hoppers of the automatic ore feeders, by which it is supplied to the stamps.

The uniform size most used is 44 in. diameter at large end, 36 in. diameter at small end, and 18 ft. long. Total weight about 19,000 lbs. Capacity is 30 to 40 tons per twenty-four hours.

Krom's Dry Kiln.—Fig. 70 illustrates the improved dry kiln designed by Mr. Krom, which also is intended for drying ores when they are to be worked by dry crushing.

The ore is brought in a dump car, M, and discharged into the ore-bin O, and fed directly from there into the rock-breaker, K, from which the broken ore falls into the hopper, *l*, which

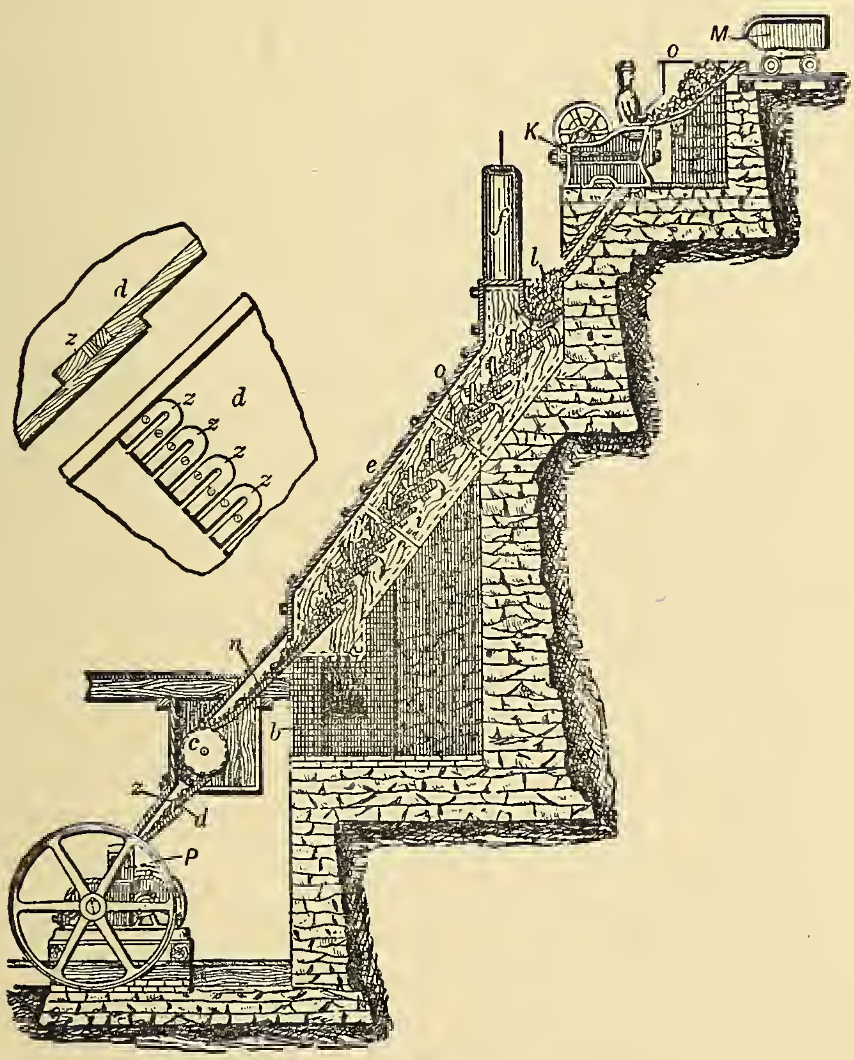


FIG. 70.—KROM'S DRY KILN.

feeds the furnace. The cast-iron plates, *b*, on which the ore rests while drying, are arranged in steps, with spaces between each step of 3 or 4 in. These spaces allow the hot air and gases from the fire underneath to pass up through the strata of coarsely crushed ore, as plainly indicated by the arrows.

The waste heat and evaporated moisture pass out through the chimney, *f*. The plates, *b*, or steps, are placed at an angle

of 45°, but it will be observed that the furnace assumes an angle of 58°; therefore, to maintain the strata of ore of a uniform thickness, it is necessary to have the check-plates, *a*. The distance between the lower edge of these plates, *a*, and the plates, *b*, determines the thickness of ore strata. This thickness can range from 6 to 9 in. The check-plates, *a*, can be readily varied in height by means of holes in the flanges which support them.

If the location of the mill site will admit of so much elevation, the plan shown in Fig. 70 will save labour and expense in handling the ore. The plan thus suggested is for continuous and automatic operation by discharging direct into the rollers. Feed roller *c* is to control and regulate the flow of ore to the crushing rollers. As it is found necessary to employ some means for collecting the pieces of iron and steel before they pass to the pulverising machines, a system of magnets is placed in the shute *a*, 2, *d*. The arrangement of the magnets is more plainly shown in *d*.

This dry kiln has decided advantages over the revolving drier. As here illustrated it requires no power to operate it, and the gentle flow of ore by gravity over the plates does not tend to stir up dust or create any by abrasion, and consequently no dust-chamber is required, as with the cylinder drier. The fuel required to dry ore in the kiln is the minimum amount. The kiln is 20 ft. long, 5 ft. wide, and holds a strata of ore 6 to 8 in. thick. The capacity may be estimated as between 2½ to 5 tons per hour.

CHAPTER X.

SUNDRY APPLIANCES FOR SILVER MILLING.

SECTIONAL MACHINERY FOR TRANSPORTATION—Amalgam Safes—Quicksilver Elevator and Tanks—Quicksilver Pumps—Krom's Improved Crushing Rolls—Specifications for Wet and Dry Crushing Mills.

Sectional Machinery.—For the purpose of mule-back transportation, crushing mills are so constructed as to be in pieces which do not exceed a certain weight, and which can be tightly fitted together on the spot. Mortars are made—as shown in Fig. 71, the base being of cast iron—in sections, planed and

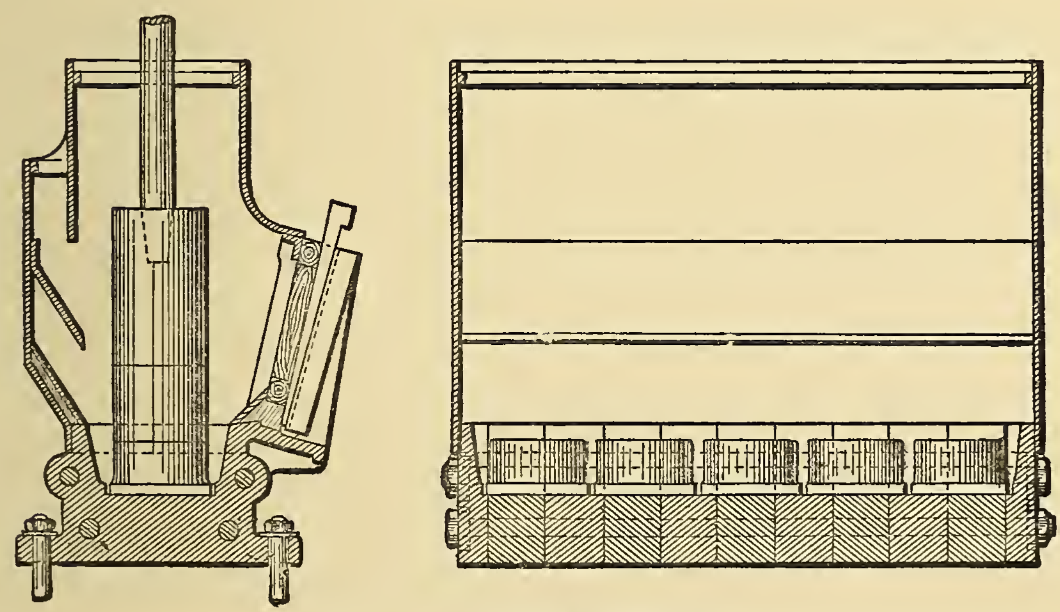


FIG. 71.—SECTIONAL MORTAR.

fitted to each other, and bound together longitudinally by perfectly fitting bolts. The housing is made of boiler iron, to reduce the weight and avoid possible breakages, and the joint is made in a peculiar manner which insures perfection.

Engines, boilers, shafting, pulleys, and bearings, stamp mills, amalgamating pans, settlers, roasting and smelting furnaces, concentrators, rock crushers, Cornish pumps, hoisting engines, can all, in like manner, be made in sections, so that no piece will exceed 300 lbs. in weight, or less if necessary.

Amalgam Safes.—In some mills the canvas bags, B, for the straining of the quicksilver are hung in boxes, D, whose lids have openings, O, just large enough to pour in the quicksilver. The appended illustrations (Fig. 72) represent a form of amalgam safe which is in common use.

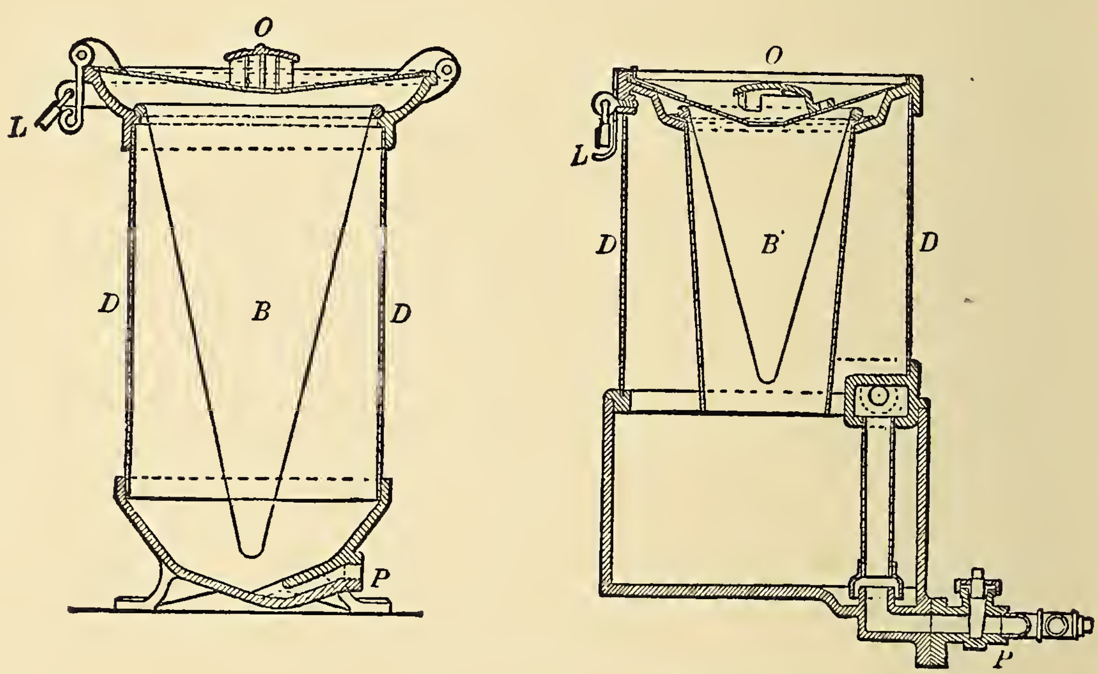


FIG. 72.—AMALGAM SAFES.

The body is of sheet iron containing a locked door, L, through which to reach the strainer or canvas bag. The top and bottom are of cast iron. A hinged cover is attached to the top and secured by a padlock, as shown. The safe is placed below the syphon discharge of the settler, and receives the amalgam and quicksilver from the settler through the opening in the cover, the canvas bag or strainer serving to filter the quicksilver-retaining amalgam in a stiff, pasty condition, and the excess of quicksilver is drawn off from the bottom in pipes, P, leading to the quicksilver elevator, by which it is returned to the upper part

of the mill building ready for redistribution. The safe is kept locked to prevent the precious amalgam from being stolen.

In some mills hydraulic strainers are employed, saving a large amount of labour and expense in retorting; but when treating "base ores," which dirty the quicksilver, it is better not to employ hydraulic strainers, so as to retort as much quicksilver as possible and get it nice and clean by distillation.

Quicksilver Elevator and Tanks.—Owing to the great mechanical loss involved in handling large quantities of quicksilver, several contrivances have been invented to remedy this evil. One of the earliest devices was the quicksilver elevator. This elevator is much the same in form as the ordinary belt elevators; but the cups, B, attached to it are made of Russia iron, and of a peculiar shape, especially adapted for carrying quicksilver. The lower pulley, P, and bearings are carried by a cast-iron boot, to which, and extending up to and around the upper pulley, O, is attached a wooden casing, C, which should be made perfectly tight on the lower side and joints, to avoid possible loss of quicksilver spilled from the cups. This casing is preferably made of iron. The upper tank, T, receives all the quicksilver,

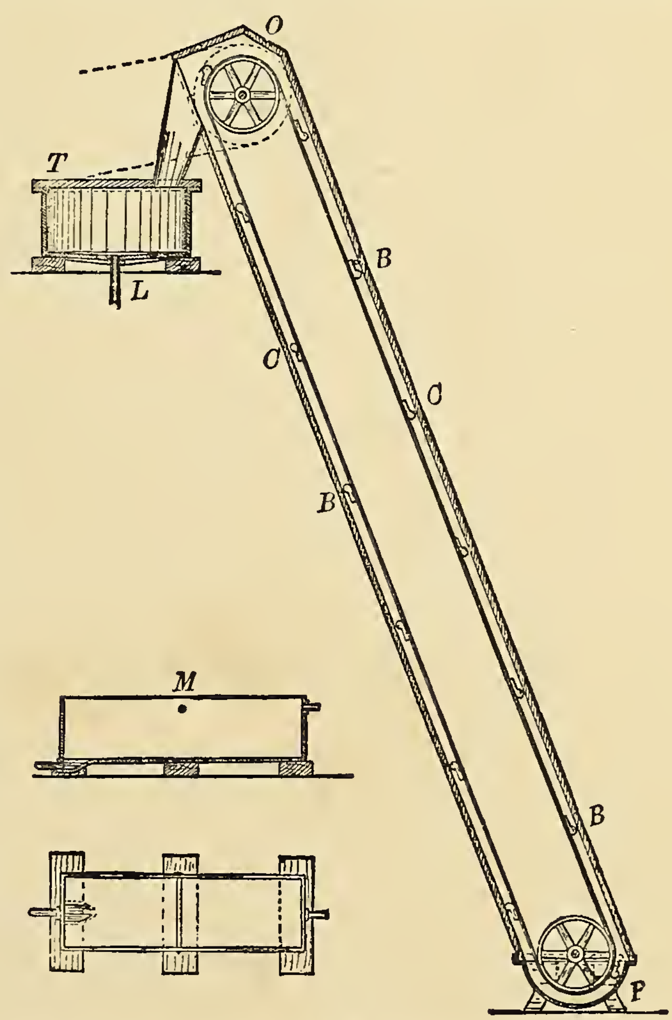


FIG. 72A.—QUICKSILVER ELEVATOR AND TANKS.

and is made of cast iron. At the bottom a pipe, L, leads off, and from this other pipes distribute the quicksilver to stamps,

pans, and settlers, as may be desired. The tank, M, shown by two views at the lower part of the illustration, is a receiving tank for quicksilver returned and to be again elevated. This tank is also made of cast iron.

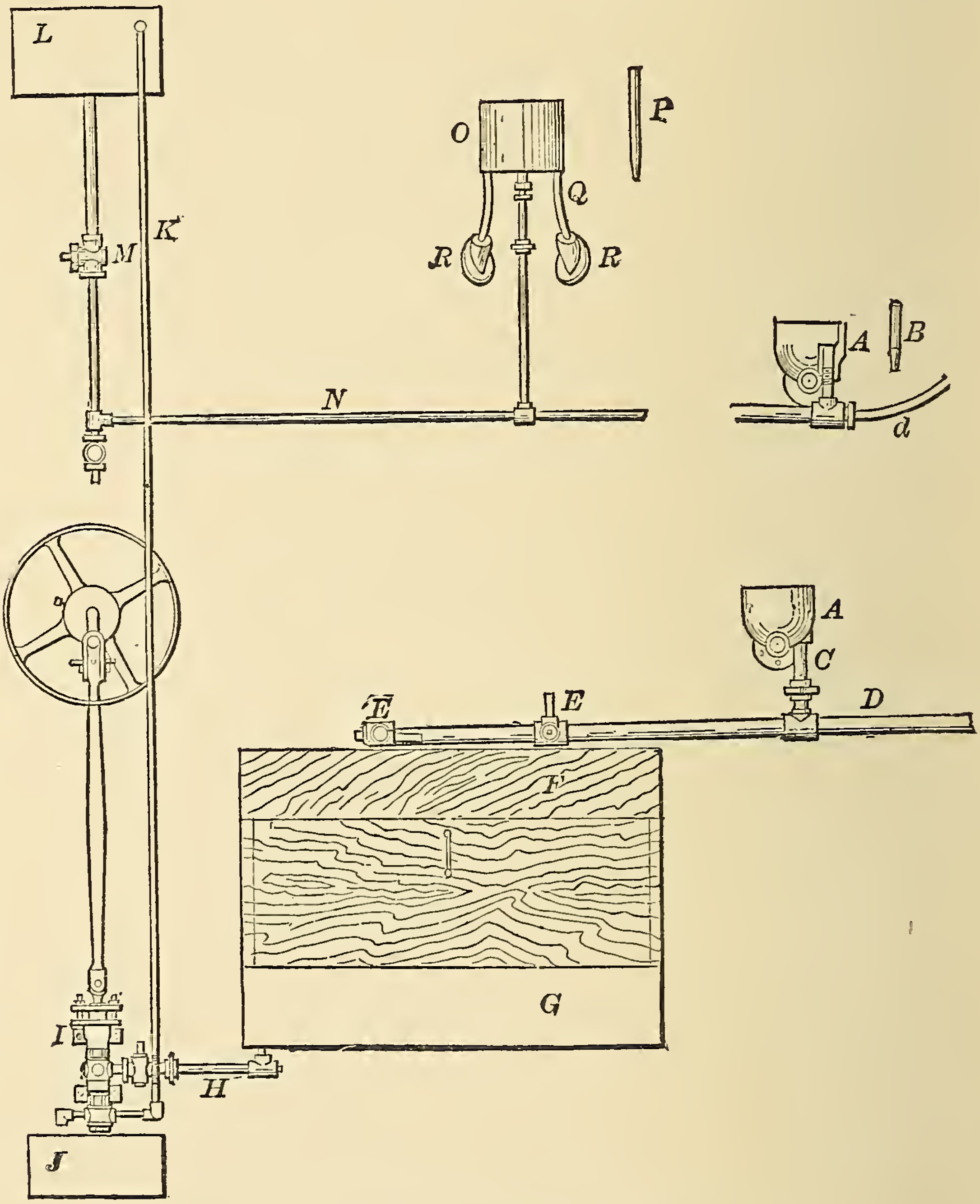


FIG. 73.—QUICKSILVER PUMP SYSTEM. Elevation.

Quicksilver Pumps.—Pumps, however, are now taking the place of the elevators for raising quicksilver. In this arrangement, the reservoir of mercury is placed in the lowest point

of the mill where mercury is used, and from this point it is lifted by a mercury pump to a tank placed at a point a little above the highest point where it is to be used, and is discharged from this tank to the various pan receivers fastened to each pan.

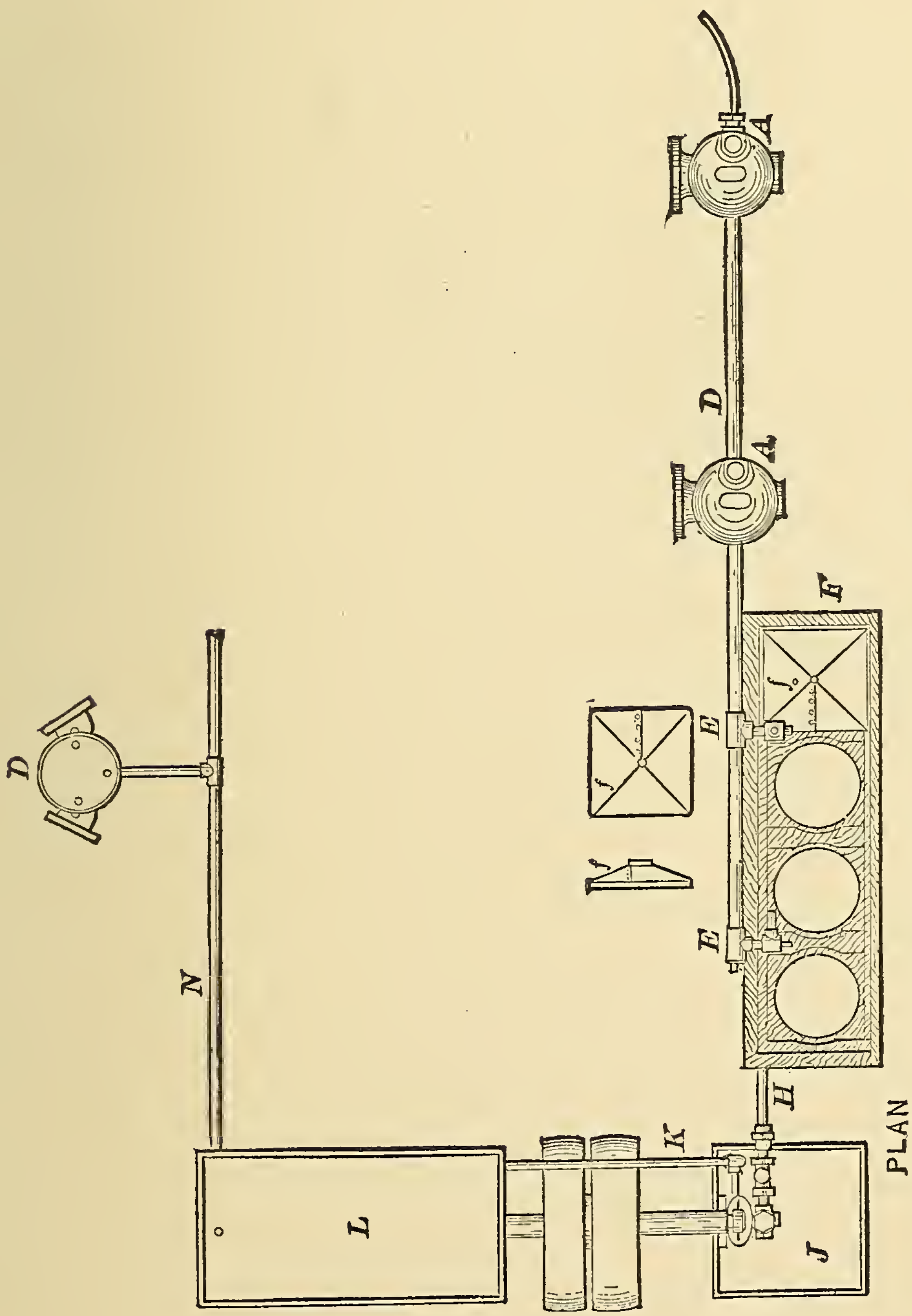


FIG. 74.—QUICKSILVER PUMP SYSTEM. Plan.

The arrangement of the pump system is shown in Fig. 73 in elevation, and in Fig. 74 in plan.

The settler bowls, A, have a vertical discharge at one side ;

said discharge is 4 in. above the inlet, and 1 in. above the bottom of settler (inside), and is fitted for a wooden plug, B. A 1-in. pipe, C, connects the discharge to a $1\frac{1}{2}$-in. pipe, D. This pipe, D, has an incline of 1 in. in 4 ft., and at the upper end a small pipe, *d*, connects with a steam pipe for occasional clearing out. At the lower end of this pipe are the strainers. Short 1-in. pipes, E, may be turned to either of two strainers, and discharge on sheet iron covers, *f*. These covers may be locked by passing a rod over them.

Strainer box, F, is of wood, and contains four strainers. It is fitted to the top of a cast-iron tank, G. Tank G is 5 ft. long and $1\frac{1}{2}$ ft. wide, and 1 ft. deep. Through the bottom of G a short 1-in. pipe, H, connects to the pump I.

The pump (Fig. 75) has a stroke of 5 in., and makes 40 revolutions per minute. The plunger is of steel, and the packing is $\frac{3}{8}$-in. round rubber, to prepare which it should be dropped into melted tallow and wiped dry before cooling; this dispenses with oil on the plunger. Around the plunger, below the stuffing-box, is a recess into which water rises, making a hydraulic packing below the rubber. Rubber gasket $\frac{1}{8}$ in. thick is used on the plugs.

A tank, J, stands under the pump to catch leakage. The pump should run only when throwing. The valves being of rubber and seating easily, and the plunger running in hydraulic packing, leaves no possibility for any grinding effect on the quicksilver.

The pump discharges through a $\frac{3}{4}$-in. pipe, K, (this pipe should be well supported near the pump), into an upper reservoir, L. This reservoir stands on a framework, 20 in. above the pans. The vertical discharge is a 1-in. pipe for 18 in. or 20 in., down to a quicksilver cock (cast iron with a stuffing-box), M. From this cock the distributing pipe, N, is $\frac{3}{4}$ in., leads in front of the pans, and may conveniently be on the floor (being protected by wooden strips on each side). At the lowest place in this pipe provision must be made to allow drainage into a settler or to the reservoir below. Between each of two pans a branch of pipe, N, enters the bottom of charging bowl,

O. Before entering the bowl the pipe is enlarged to 1 in., to accommodate a better size of wooden plug, P.

Charger O is of cast iron, stands 7 in. above the top of the pans, and is supported by two stand pipes, Q (1 in.), which connect to the pans by flanges R. These flanges, for side iron pans, are bolted to the outside, and stand at an angle of 60° or more.

A belt shifter for starting or stopping the pump should be operated from near the upper reservoir.

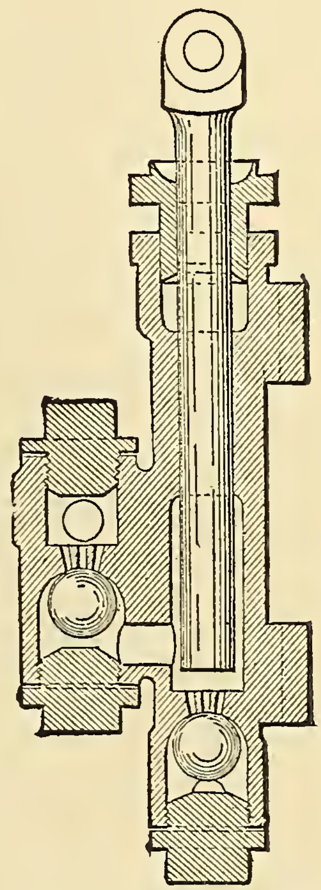


FIG. 75.—QUICKSILVER PUMP IN SECTION.

Krom's Improved Crushing Rolls.—Since the introduction of leaching processes for the extraction of silver (see *post*, Ch. XIII.) and their manifest success, it has been found that the mechanical preparation of the ore by the stamping mill is not adapted for the leaching method, as too much fine dust is produced. Experiments carried out with crushing rolls showed that the resulting pulp was much more uniform in size, and better adapted for the liquids to pass through it. There was also an impression among western mill-men that the finer the ore was crushed the better it was adapted for chloridizing roasting, and that in wet amalgamation the mercury could attack the gold and silver better if the pulp were very fine. Practical experience, however, has demonstrated that for chloridizing roasting excessive fineness of ore is not necessary, and that it may be even injurious in lixiviation, by interfering with the rapidity and efficiency of the filtration.

The crushing capacity of Krom's rolls is very large when compared with a stamping mill. Two sets of rolls, of which one set is 14 in. long by 26 in. in diameter, and the other set 16 in. by 30 in., are equal in capacity—at a velocity of 100 revolutions per minute—to a 50-stamp mill.

Favourable reports from various competent mining superintendents as to the efficiency of Krom's rolls induce me to insert a description of them in this book, as no doubt any machinery which will reduce the cost of an extensive plant like that of a large stamping battery is of interest to the mining community at large; and any invention of new processes, whereby economy results, is of importance to the working of low grade

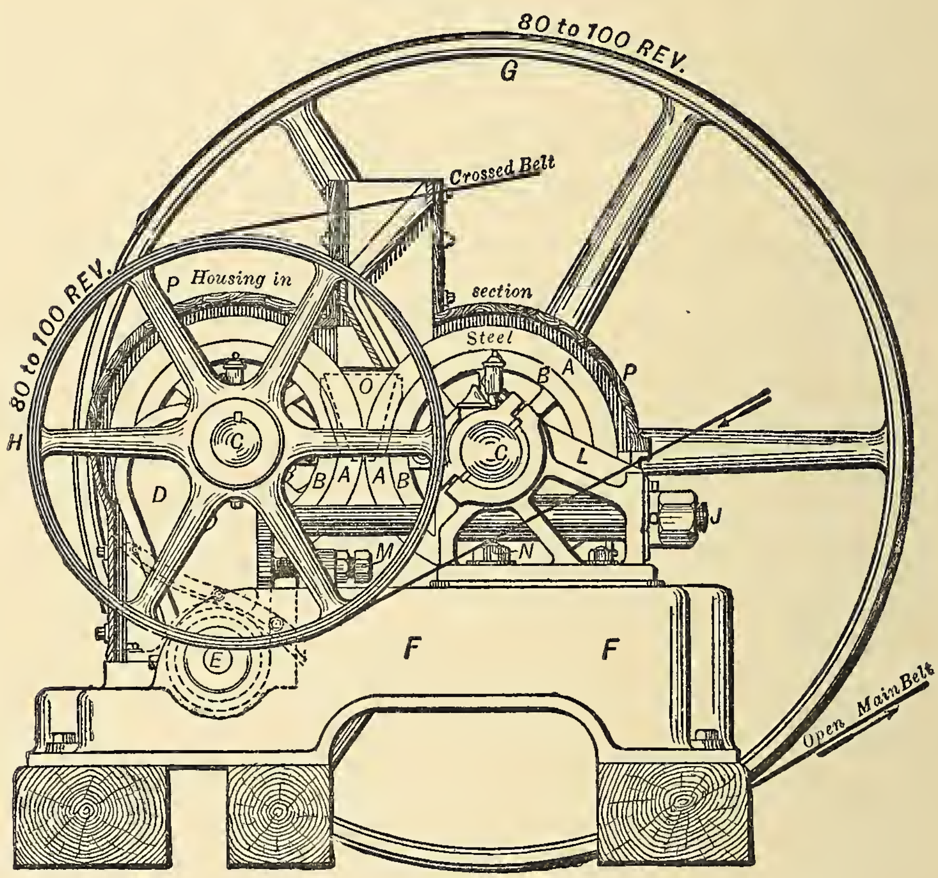


FIG. 76.—SIDE VIEW OF KROM'S IMPROVED STEEL CRUSHING ROLLS.

ores and the opening of new mines, for large sums of money are not always at the command of the prospector and miner.

The Krom rolls (shown in Figs. 76 and 77) are composed of two sets of cast-iron cores, B, on steel axes, C C′, carrying steel tyres, A. The rolls vary from 26 in. to 30 in. in diameter, including the steel tyres. These tyres are made of the best open-hearth steel, and are $2\frac{1}{2}$ in. thick on the 26-in. rolls, and $2\frac{3}{4}$ in. thick on the 30-in., and can be worn down to $\frac{1}{2}$ in.,

having (it is said) been worn as low as $\frac{1}{4}$ in. They are used until they are so thin that they become loose from expansion.

Subjoined is the detailed account of the appliance which has been supplied to me :—

"The pillow-blocks, L, of one of these rolls is firmly bolted to the bed-plate, F, by the nut, N. The second one is set in a swinging pillow-block fixed in two strong cranks, D D′, which rotate in a journal, E, set in the cast-iron frame, F. The shaft, *m*,

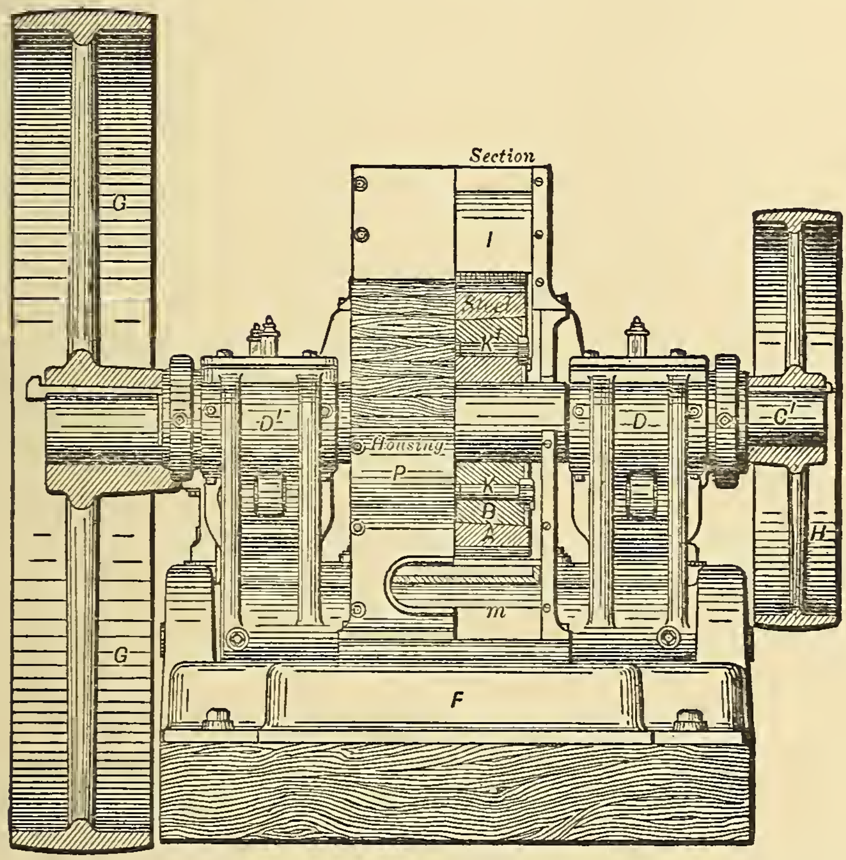


FIG. 77.—END VIEW OF KROM'S IMPROVED STEEL CRUSHING ROLLS.

which connects the two swinging pillow-blocks, is 11 in. in diameter, so that this roll and shaft are always held in line with the other roll. When rolls are constructed with pillow-blocks which slide on the bed-plate independent of each other, it is difficult to keep the two shafts parallel ; besides, there was always a liability of the movable pillow-blocks getting loose on the bed, which is not only objectionable, but causes damage to the bottom face of the pillow-block and the face of the bed-plate. But with this construction all the pillow-blocks are

securely fixed. One pair is firmly bolted to the bed, while the movable pillow-blocks swing on the strong pivot, E. The distance between the two rolls is regulated by the screws, M, one on each side, with their jam nuts to prevent any motion after they are once set. The rolls are held by two heavy bolts, J, so that the position once fixed the distance of the surfaces cannot be changed by any action of the machine, but only by the wear of the rolls. To the fixed rolls a large driving wheel, G, 7 ft. in diameter and 15 in. wide, is keyed on its axis, C, and to the movable one a smaller wheel, H, 42 in. in diameter and 8 in.

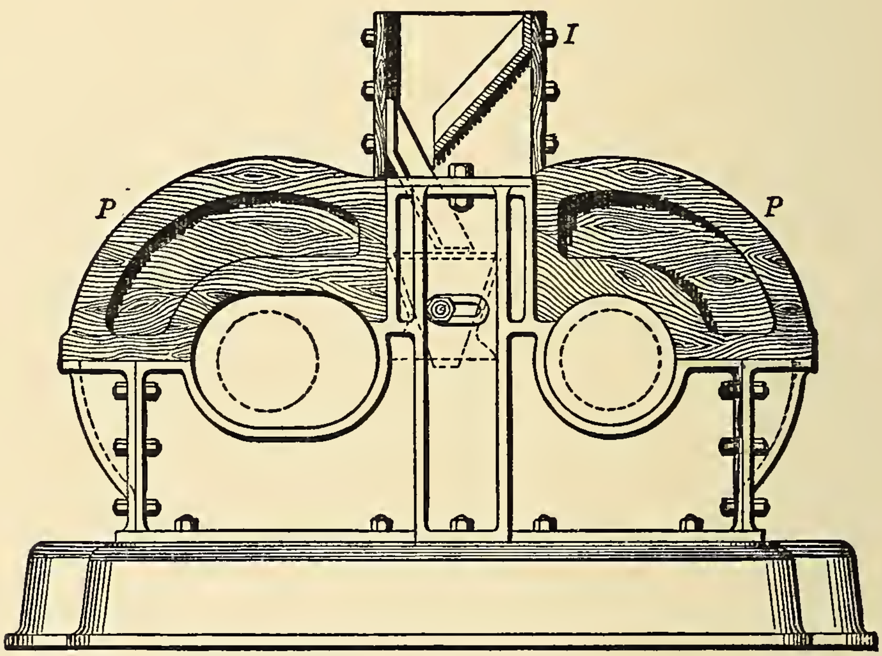


FIG. 78.—HOUSING FOR KROM'S PATENT STEEL CRUSHING ROLLS.

wide, is keyed on its axis, C′. The rolls are covered with a housing, P, shown also in Fig. 78, to which an exhaust fan is attached, so that no dust escapes into the air, the whole of it being carried to the dust chamber. To this housing a feed box, I, is attached, which is provided with a series of inclines, so as to spread the ore in a continuous and even sheet between the surface of the rolls. Magnets are attached to the feed chute to catch any pieces of hard steel which may have fallen into the ore from broken tools, which might dent the surface of the rolls; iron will do no injury to the faces.

"The machine is very compact, a pair of 26-in. rolls occupy-

ing a ground space of 7 ft. by 7½ ft. The tyres, which for the 26-in. roll weigh 816 lbs. each, are held in place by two cast-iron heads, B, which are slightly conical in shape. One of these is shrunk on the shaft, the other is slit on one side and slips on to it. Both of the heads, B, are so placed on the shaft that the smaller diameter will be toward the centre. The steel tyre is turned out on the inside to correspond to this, so that it can be easily slipped over the permanent head and the loose core brought up to it. The two are securely fastened together by bolts, K, so that when the movable head is drawn up to the permanent one the slit in it closes and makes it perfectly tight on the axle.

"It was thought at first that the steel tyres would be found to wear unevenly, but they do not. When the tyres are worn thin they become loose; they can then be very easily removed and others substituted. To prevent loss of time it is a good plan to have duplicate rolls, so that there may be no delay when the tyres are worn out, as the putting on of a new set requires but a short time. Movable check pieces are placed at each end of the rolls to keep the ore from spreading sideways. These check pieces are also adjustable.

"The rolls are usually arranged in sets of two, or, when very fine crushing is intended, in sets of three."

Specifications for Wet and Dry-Crushing Mills.—The following specifications for ten and twenty-stamp silver mills may be usefully given here:—

Specification for Ten-Stamp Wet-Crushing Silver Mill.

One No. 2 Blake crusher, 10 in. by 7 in.

One grizzly or ore screen, 4 ft. by 10 ft.

Two Tulloch automatic ore feeders.

Ten stamps of 850 lbs. each in one battery, including all iron work, wooden pulley, and hardwood guides for stamp stems.

One set of water-pipes for battery.

Six combination amalgamating pans, 5 ft. diameter, complete.

Three combination settlers, 8 ft. diameter, complete.

One clean-up pan, 4 ft. diameter, complete.

One set of water-pipes, with hose for pans and settlers.

One set of steam-pipes from boiler to pans.

One china pump, to return surplus water from settling tanks to supply reservoir.

One automatic quicksilver system complete, including quicksilver elevator, distributing tank, bowls, and pipes.

Three amalgam safes.

One amalgam car.

One retort, 12 in. by 4 ft., complete with smoke stack.

One sheet-iron plate, for retort-room floor.

One bullion melting furnace, with crucible tongs and bullion mould.

One overhead crawl and one-ton differential pulley block, with chain to be placed over stamps.

One overhead crawl and two-tons differential pulley blocks, with chain to be placed over pans.

All track iron and nails for crawls.

One line of shafting running under pans and coupled to engine, with bearings and pulleys.

One countershaft for crusher, with bearings and pulleys.

All belting and lace leather.

One Corliss engine, 14 in. by 42 in.

One tubular boiler, 54 in. by 16 ft. complete.

One No. 3 steam feed-pump.

One tubular heater, 20 in. diameter.

Power required for this mill would be 65-horse. A similar dry-crushing mill would require only 45-horse, as he pans do not require so much power as in wet-crushing.

Specification for Twenty-Stamp Dry-Crushing Silver Mill.

One No. 2 Blake crusher, 10 in. by 7 in.

One grizzly or ore screen, 4 ft. by 10 ft.

Five Tulloch automatic ore feeders—one intended or revolving dryer.

One automatic ore-drying cylinder, 44 in. by 36 in. by 18 ft. complete.

Twenty stamps of 850 lbs. each in two batteries, including all iron work, wooden pulleys, and hardwood guide-boxes for stamp stems.

Four screw conveyors for front and back of mortar to convey pulp to elevator.

One elevator with belt, elevator cups, pulleys and shafts.

One screw conveyor, overhead from elevator to furnace hopper.

One roasting furnace.

Iron ore car and track iron with nails.

Eight amalgamating pans, 5 ft. diameter.

Four settlers or separators.

Two agitators.

One clean-up pan, 4 ft. diameter.

Complete set of water and steam pipes.

One line shaft running under pans and coupled to engine, with bearings and pulleys to drive pans and settlers.

One countershaft for agitators.

One countershaft for stamps, with bearings and pulleys.

One countershaft for crusher and ore dryer, with bearings and pulleys.

One countershaft for roasting furnace, with bearings and pulleys.

All belting and lace leather.

Two amalgam retorts, 12 in. by 4 ft. complete, with smoke stack.

One plate for retort-room.

Two bullion melting furnaces, with crucible tongs and bullion moulds.

One overhead crawl and one-ton differential pulley block, with chain to be placed over stamps.

One overhead crawl of two tons capacity to be placed over pans.

One Corliss engine, 18 in. by 42 in.

Two tubular boilers, 54 in. by 16 ft., complete with steam drum.

One steam feed-pump, No. 4.

One tubular heater, 24 in. diameter.

Power required for a Twenty-Stamp Dry-Crushing Silver Mill.

One Blake rock-breaker, No. 2 ...	6	horse-power.
Five ore feeders	0	„
Twenty stamps of 850 lbs., 80 drops	25	„
One revolving ore dryer	5	„
Two Brückner cylinders	10	„
Eight amalgamating pans, 5 ft. diameter	16	„
Four settlers, 8 ft. diameter ...	12	„
Two agitators	4	„
Friction	20	„
Total	98	„

CHAPTER XI.

CONCENTRATION AND ORE DRESSING: SIZING.

ORES AS THEY COME FROM THE MINE—Separation of the Ore—The Theory of Separation—Sizing Apparatus—Coarse and Fine Concentration—Sizing of equal-falling Particles—Continuous Cylindrical Trommels—Single Conical Trommel—Revolving Screens—Separation without Sizing—Pyramidal Troughs—Triangular Double Troughs—Conical Apparatus—Bilharz Spitzgerinne.

Constituents of Ores.—Ores as taken from the mine always consist of a *valuable* and a *worthless* portion; and there may also be an *injurious* portion, involving loss in the subsequent treatment. It very often becomes an economical question whether the worthless and inert, and the active and injurious, portions should be removed by mechanical means before the ores are subjected either to the smelting process in furnaces or to the amalgamation process. Very often the tailings contain sufficient valuable mineral which, if separated from the large quantity of worthless gangue accompanying it, can be profitably reduced, as is being done now in several establishments.

There are localities where silver ores are found of sufficient richness to be smelted without previous preparation, but instances in which a proper dressing would not be advantageous are rare. Even in rich ores there is usually much more earthy matter than is necessary for the formation of sufficient slag to protect the metal in the furnace, and the removal of this excess by dressing before the ore comes to the furnace would always be found economical. The same remark applies to a proper system of amalgamation.

By means of a rational and comprehensive system of dressing,

however, the galena, the pyrites, and the blende, which may have been intimately associated in the ore as it came from the mine, may be separated cleanly enough for all practical purposes, and each subsequently treated for itself.

In all mining operations it is clear that when the ores are taken out of the mine the earthy portions of the ore must be got rid of before we can get the metal itself. The economical question is, whether it is cheaper to wash away a considerable portion of this dead mass or to pass it through the mill or the furnace along with the productive portion.

Any concentration will remove a portion of the unproductive part, and no economical system will perfectly separate every trace of it. We are compelled, therefore, to resort to a partial separation if we resort to any, and the question how far it will pay to go becomes the important one. It would not usually be desirable to separate every trace of gangue from the ore even if we could. If the ore is to be transported for great distances the cost of transporting worthless material should be taken into account; but under other circumstances it is not desirable to pass a certain limit in the concentration, which must be determined as before by actual study and experiment.

In some instances it has been found profitable to dress and concentrate ores which have to be amalgamated, or roasted and amalgamated, but it has to be determined first if the loss in concentration would be greater than the cost of passing the inert matter through the roasting furnace. Tailings are generally concentrated now before being submitted to amalgamation.

The Theory of Separation.—Gaetzschmann and Rittinger, whose works on the "Aufbereitungskunde" (the "Science of Ore Dressing") are so well known, lay down the following rules on this subject:—

The best possible conditions for dressing ore would be to have the valuable portions of the ore possessed of a specific gravity greater than the liquid in which the dressing is to be performed, and the worthless portion of a specific gravity less than that of the liquid. For example, let us suppose that an

ore consisting of galena, the specific gravity of which is 7·5, and quartz of the specific gravity 2·5, so finely crushed that each particle consisted of one of the minerals alone, to be located first in a liquid of a specific gravity 5. It is evident that the quartz would remain floating on the surface, while the galena would be found at the bottom of the vessel containing the liquid. Practically, however, it would be found that pieces consisting partly of galena and partly of quartz—whose specific gravity was less than that of the liquid—would remain floating, because it would be practically impossible to separate the minerals entirely by crushing, and fulfil one condition of our problem. It is evident, therefore, that if, even in the most favourable conditions we can possibly imagine, the mechanical mixture of the ore is such that we can never divide it into pieces consisting entirely of one mineral, we cannot hope, by the ordinary appliances for dressing ore, to be able to achieve perfect results. Worthless and valuable minerals can never be entirely separated when they occur mixed together in an ore.

We may, however, by passing an ore once through a single machine, obtain one portion which shall contain, practically, none of the accompanying gangue, but we must do it at the loss of a considerable portion of valuable material; or we may obtain a portion which shall, to all intents and purposes, be free from the valuable constituent of the ore, but this can only be done by leaving a considerable portion of worthless material mixed with the valuable portion. It is not more difficult to make the separation in one way than in the other, but the question of economy must be again considered in determining which of the two limits we will most nearly approach. When the ore is very valuable for the precious metal it contains, and at the same time consists mainly of gangue, it would probably be found advantageous to remove only so much of the worthless portion as could be separated with but very slight loss of the valuable mineral.

The question of an economical concentration where the ore is to be separated into only two portions is much less compli-

cated than that of dressing an ore composed of several minerals, which it is desirable to separate as cleanly as possible.

Ore cannot be dressed except in some medium which offers a resistance to the force of gravity. Water is the most convenient medium in which the separation of ores can be made.

Since all ore dressing is founded on the different behaviour of substances of different specific gravities, the study of the action of the laws of gravity will not be out of place here.

If a mixture of particles of different sizes and different specific gravities were allowed to fall a certain distance through water (supposing all the particles to start together), they would form a deposit on the bottom of the vessel, separate horizontal layers of which would consist of particles which reached the bottom at about the same time. It is evident that the undermost layer would consist of coarse and very dense particles, together with coarse particles of less specific gravity. The uppermost layer would consist of the finest particles of the densest substances, with larger particles of the less dense substances. If each layer is separately removed, the different sized particles can be easily separated by sieves of different sized meshes into parcels of the same, or nearly the same, specific gravity. The smallest grains will, of course, be the densest, and the largest the least dense, in each layer.

If, however, the separation of the grains, according to size, be undertaken previously, and only sized material be thrown into the water, the layer of the deposit at the bottom will be composed of particles of the same specific gravity. It is important, therefore, that the laws which govern the falling in water of bodies of different sizes and densities should be well understood, for upon them is founded the practice of wet dressing of ores.

The action of the water is to afford a resistance to the operation of the force of gravity, and to cause bodies heavier than water to fall more slowly through it than they would through air.

This effect may be augmented by placing the bodies in a column of water which is in motion upward. The fall of a

body would then be opposed by a force arising from the positive motion of the water in the contrary direction, which must be added to the resistance due to the density of the water in order to find the total resistance to gravity.

To do good work in concentrating ore a previous accurate sizing of the stuff to be treated is necessary.

The Sizing of Equal Falling Particles.—The separation of equal falling grains according to specific gravity is, of course, a simple sizing, and can be effected by the use of sieves, when the grains are of sufficient size to permit treatment. In the practical dressing of ores, however, only those portions of the ore which are too fine to be sifted to advantage are first separated into equal falling grains, and subsequently according to specific gravity. Previous sizing has been found more economical, because the sieve is the most simple and rapid contrivance to effect the result, and because sized ore can be separated according to specific gravity more easily, economically, and rapidly, than equal falling grains. If sizing by means of sieves can be effected after the separation according to the falling power of the grains, it could have been effected to better advantage beforehand. Of equal falling grains the smallest are of course the densest, and the largest the least dense. The smaller grains will be mainly ore, the larger mainly gangue.

A very thin smooth stream of water, passing slowly over a plane surface exerts different forces upon large and small grains lying in the current. The highest points of the small ones lie very close to the plane surface. The friction on the layer of water next to the surface over which it runs is much greater than on the layer above, so that very small grains which lie wholly in the lowest layer will be much less acted on by the force of the current than larger grains, the tops of which protrude into the layer above. In accordance with this principle numerous machines have been constructed for separating equal falling grains according to size, and hence according to their specific gravities. The thinness of the stream is a necessary condition for the successful operation of any of these machines

for the purpose for which they are intended. A thick or deep stream acts upon all points very nearly equally, so that the small particles are propelled forward with nearly the same velocity as larger ones. In such a stream a sizing of grains would be impossible.

Another requisite condition for the success of this method of sizing is that the stream shall have the right velocity, which depends upon the inclination of the plane over which it flows. If the plane is too nearly horizontal the force of the current will not be sufficient to carry off even the coarser particles, and if it is too steeply inclined the force will be sufficient to sweep away fine and coarse particles alike. The proper inclination of the plane is, however, not difficult to discover in practice, and when the angle can be altered by means of a screw or lever, the proper adjustment is easily made.

The number of grains held in suspension in a given quantity of the water allowed to run on a plane—or, in other words, the muddiness of the stream—must also be regulated. If the water is too muddy it will not be free to act on the separate grains in the manner described above. The grains will act on each other, and the resultant force acting on a given grain will be composed of the force of the water current acting on the grain, and the force imparted by the grains lying around it. It may seem a good rule to follow in working such machines to have the water as clear as possible, since the sizing of the grains would then be most nearly complete; but on the other hand less work would be done by the machine than if the water were muddier. The economical medium will be found by practical experiments.

Sizing Apparatus: Trommels.—When the velocities of the fall of grains of stuff in water are materially different, then the number of sizing and classifying divisions may be few; but when the specific gravities of several veinstones are closely allied to each other, then the classifying arrangements must include a greater number of subdivisions.

The separation of stuff into grains of different sizes can be

effected by a set of revolving trommels, or sieves. Such an arrangement is only applicable where a coarse concentration is required. *Coarse concentration* means where the ore is crushed coarsely, divided into several classes or sizes, and treated on various different concentrating machines; *fine concentration*, in which the ore is crushed finely and treated without classification on one style of concentrating machine.

A combination system is occasionally employed on some ores, in which the material is crushed finely but classified into two or more sizes, being treated on two styles of concentrating machines, or treated separately on machines of one style differently adjusted.

The ores adapted to coarse concentration are those containing the mineral to be saved in large crystals, masses, or seams, so that when broken in comparatively large pieces a good separation is effected between waste rock and valuable mineral. Many ores of lead, zinc, copper, and iron are of this character. The ores adapted to fine concentration, on the contrary, contain the valuable mineral in fine particles or crystals disseminated through the mass of the rock in such manner that a coarse crushing would leave the pieces of waste rock still impregnated with mineral, and a finer crushing therefore essential. Ores of silver, gold, and tin are usually of this character, the silver ores frequently comprising also combinations of lead, copper, and zinc, of secondary importance.

To effect the sizing of the stuff, continuous cylindrical trommels, revolving on one common axis, are employed.

In order to effect a thorough division of the grains by means of revolving trommels, water should freely enter into and fall upon the cylinders. The crushed stuff should therefore drop into a stream of water, and water should also be delivered to the outside of the trommels from distributing launders or pipes.

Cylindrical sizing trommels require to be set at angles varying from 3° to 5°, in order to pass the stuff from the entering to the discharging end. The axis of the conical shaped trommel may be strictly level, since the falling angle of the perforated shell, usually about 3°, combined with its rotative

movement, will suffice to impel the stuff from the smaller to the larger end.

This trommel is shown in Fig. 79. The stuff entering at the farther end from the driving rigger passes to the ½-millimetre section, then successively over the ¾, 1¼, 2, and 3 millimetres divisions. The thin plates are supported by lateral bars fastened to the rings within and without the cylinder, one of such bars being shown. Wrought-iron rings, about 1½ in. wide and ¼ in. thick, divide the cylinder into sections, and serve as fasteners for the plates and for connecting the radial arms proceeding from the central bosses. The order in which the sizing is effected in this trommel sieve is from small to large grains, consequently the initial quantity of stuff passes

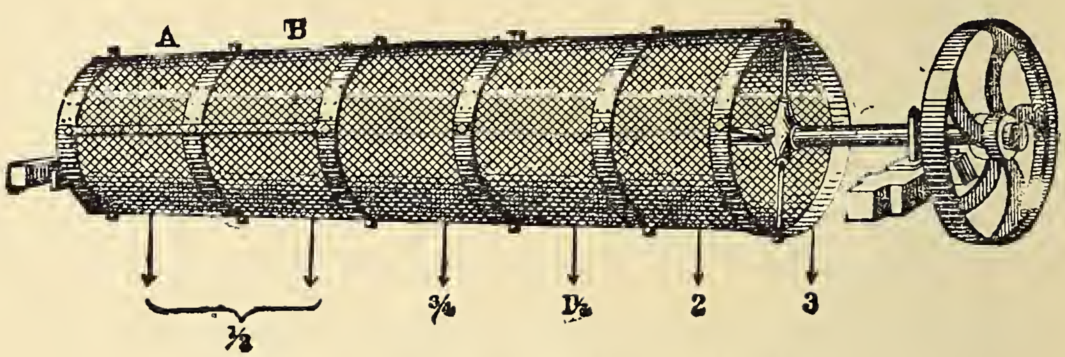


Fig. 79.—Continuous Cylindrical Trommel.

first over the thinnest plate, and, in the absence of careful feeding, the fine ½-millimetre holes may be overcharged, when an imperfect sizing will be the result; on the other hand, the separation being performed on a continuous line without a material loss of fall, this trommel is a useful one for many dressing-floors. The diameter of these trommels run from 2 to 3 ft., the number of revolutions per minute from 18 to 25.

The Single Conical Trommel (shown in Fig. 80) may form one of a set for sizing stuff. The order in which the sizing occurs will be from great to small grains, hence the thinnest plate and smallest holes will be subjected to a minimum amount of wear. A system of six trommel sieves, each 42 in. long, diameter at larger end 21 in., and at small end 18 in., making

20 revolutions per minute, will size from 20 to 30 tons in ten hours. The water required for the six trommels will be from 12 to 15 gallons per minute. In order to convey the stuff from one trommel to another, a sheet iron or wooden hopper may be employed. When a set is fixed step like, one above another, on a running angle of say 40°, the driving gear may consist of a shaft and bevel wheels ; but the necessary motion may also be communicated by a light side rod and cranks, in connection with a small fly-wheel.

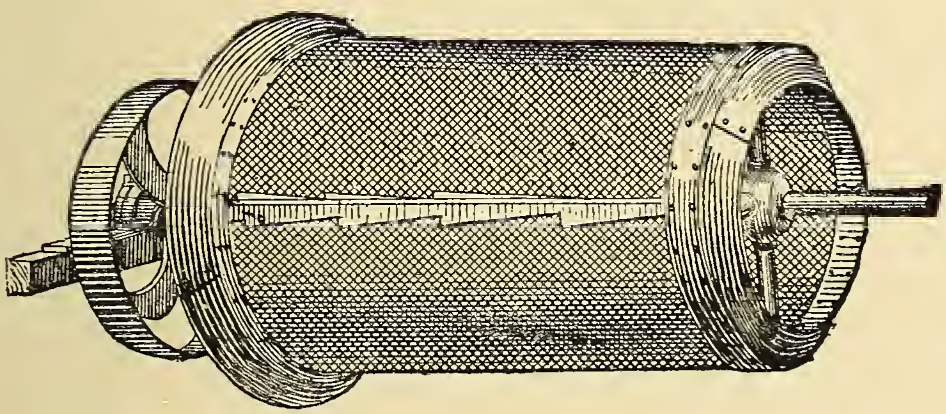


FIG. 80.—SINGLE CONICAL TROMMEL.

Revolving Screens.—In Fig. 81, the fine material from one screen passes to a second finer screen, and so on to the required number. The material remaining on each screen, and afterwards discharged to the proper jig, is thus sized.

Separation without Sizing.—When very fine stuff is to be dressed, it is found impracticable to size it accurately and rapidly upon sieves or trommels ; and for this reason an entirely different principle must be employed for treating it. It must, in the first place, be separated into *equal falling* portions, the grains composing each portion being of such relative size and specific gravity that they will sink through a column of water of a given height in equal times. Each of these portions is then treated alone upon a machine capable of separating the particles according to their specific gravity.

Thus we see that, according to circumstances, ore may be dressed by a sizing operation, and a subsequent separation of

the particles according to their specific gravity; or the separation according to specific gravity, may be undertaken after a separation into portions consisting of equal falling grains. In any case two distinct operations are necessary, and any method of dressing ores consisting of but one machine and one operation cannot possibly give as good results as machines worked upon long known and thoroughly tested scientific principles.

When the stuff to be worked is not too fine to be separated on sieves or trommels there can be no doubt that the most economical method is to size it first, and then separate it according to specific gravity.

Sometimes, however, the nature of the ore is such that it

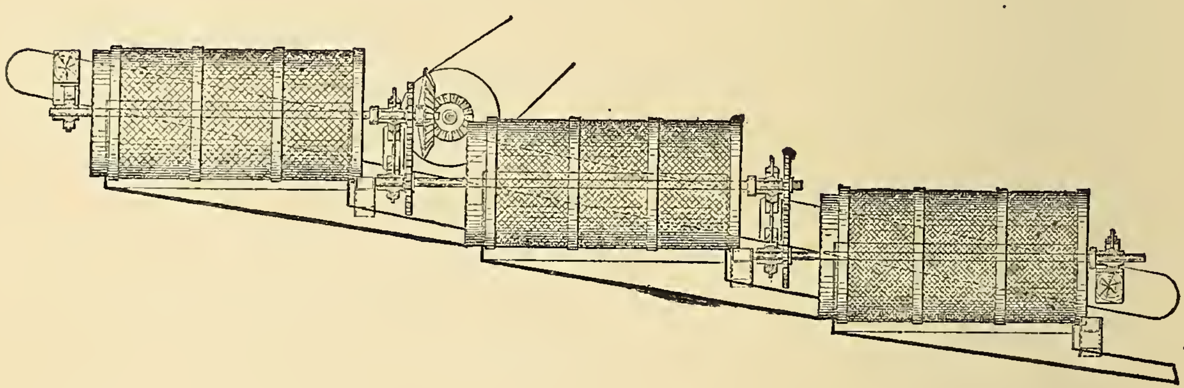


FIG. 81.—REVOLVING SCREENS.

must be very minutely crushed in order to separate the valuable portion from the worthless gangue, or a considerable amount of fine "slimes" may be incidentally produced. In such cases the second system of dressing must be resorted to, and the fine stuff, mixed with sufficient water to hold the particles freely in solution, is worked upon one of the three following principles, in order to separate and classify the *equal falling* particles:—

1st. In a horizontal stream of water of decreasing rapidity of current, and of sufficient depth, the heavier particles will first sink to the bottom, then lighter particles, and if the rapidity of the stream is properly regulated the finest slimes will settle in the almost still water at the end of the apparatus.

2nd. A vertically-ascending column of water of decreasing

rapidity of current may be so regulated that it will carry, at first, all but the heaviest particles with it; afterwards lighter particles will come to rest, and finally even the lightest will be able to overcome the action of the current, and by this means the different sorts may be separated.

3rd. A comparatively shallow smooth stream of water will allow the heavier particles to rest on the bottom of the trough, and as the rapidity of the current decreases the lighter particles will also come to rest.

On each of these principles various kinds of apparatus have been constructed. The most economical in point of labour is one constructed on the first mentioned, and called in German *Spitzkasten.*

Pyramidal Troughs (Figs. 82, 83, 84), as their name implies, are hollow, rectangular, pyramidal boxes, usually con-

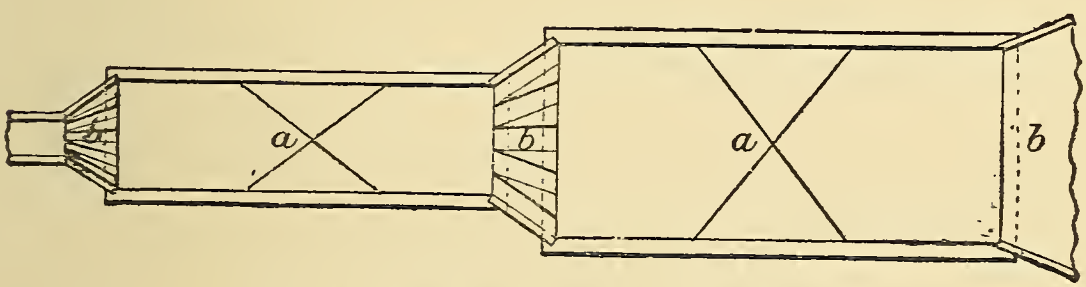


FIG. 82.—PYRAMIDAL TROUGH. Top View.

structed of strong boards well joined together. The sides are inclined at angles of not less than 50°, and there is a small hole in one side close to the apex. They are fixed horizontally in an inverted position, and the crushed material is introduced at one of the narrow ends, a few inches below the top, by means of a launder.

The result is that, as soon as the box is filled, a certain portion of the crushed matter—namely, the coarsest and heaviest, which the water on account of its diminished velocity is not able to carry with it farther—sinks and slides down the inclined sides of the pyramid and escapes through the small hole, *a*, near the apex, whilst the finer and lighter matter passes off at

the top by an outlet, *b*, in the centre of the end opposite to the point of entrance. If now a second larger box be attached to the first, a third still larger one to the second, and so on, each succeeding box at a slightly lower level, in order to prevent any settlement of stuff in the passage-ways, it follows not only

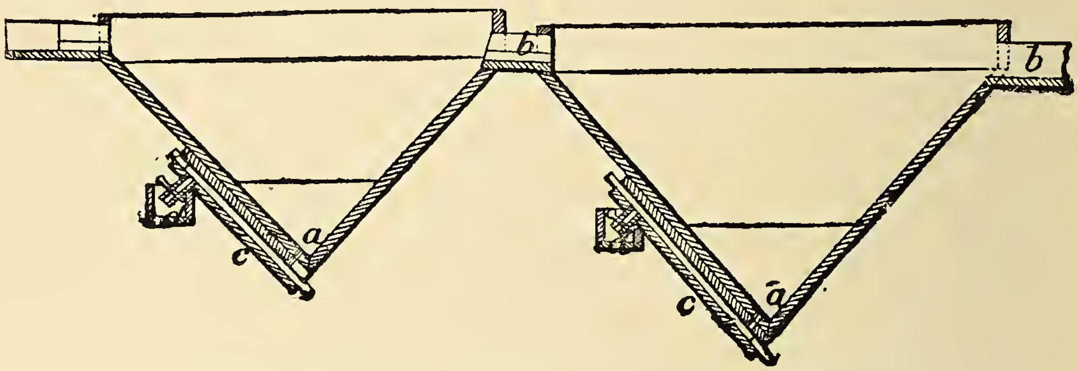


FIG. 83.—PYRAMIDAL TROUGH. Longitudinal Section.

that the same process of settling and escaping of the particles from the apex will take place in every box, but also that their size will decrease nearly in inverse proportion as the surface of a succeeding box is larger than that of the preceding one, or directly as the velocity of the water is diminished in it.

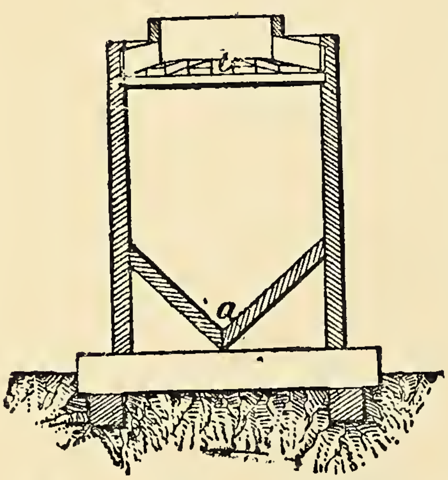


FIG. 84.—PYRAMIDAL TROUGH. Cross Section.

According to this principle of the boxes, if they were made of only very gradually increasing size, and the apex holes proportionately small, it would be possible to classify the stuff into several sets of equivalents before it entirely settled—*i.e.*, till clear water passed off from the last box. Experience has, however, shown that for fine ore dressing in general, classification depends both on the amount of material which has to pass through them per second and the size and character of the grains; and by theory and practice it has been found that for the supply of every cubic foot of material the width of the first or smallest box must be

one-tenth of a foot—for instance, for 20 cubic feet 2 feet—and for every succeeding box it ought to be about double that of the preceding one; or, generally, the widths of the boxes must increase nearly in geometrical progression, 2, 4, 8, &c., and their lengths in an arithmetical one, 3, 6, 9, &c.

Their depths depend on the angle of inclination of the sides, which, as already stated, is generally 50°, because, if less, the stuff would be liable to settle firmly and choke the central orifice, and, if larger, unnecessary greater height of the boxes would be required. The form of the two smaller boxes is commonly such that the two short sides are inclined at the above angle, and the two long ones, which would become far steeper, are broken, *i.e.*, are for a certain depth from the top vertical, and afterwards inclined at the normal angle. This modification has, however, no influence upon the action of the boxes, but simply facilitates somewhat their construction and firm fixing. The sides of the larger boxes are generally even throughout.

The way in which the outlet holes, *a*, at the apices are constructed has an important bearing on the operation of the boxes. At these points the hydrostatic pressure is considerable, and the holes would naturally be kept small in order to prevent too much water passing with the particles of stuff; such small outlets are, however, especially in the treatment of coarser material, very liable to become choked, This difficulty has been met by the holes being made of conveniently large size, but connected with pipes, *c*, $\frac{3}{4}$ in. in diameter, which rise up the side of the boxes—*i.e.* of the smallest box to within 3 to $3\frac{1}{2}$ ft., and of the others to within 2 to $2\frac{1}{2}$ ft. from the top—and are there furnished with small mouthpieces, *d*, supplied with taps for regulating the outflow. This arrangement, on account of the outlets being so much higher, has the further advantage that a considerable amount of fall is gained which for the subsequent treatment of the material is in some cases of special value.

There are two more points to which attention is requisite in order to insure good action of the apparatus—namely, the introduction of the material into the different boxes equally and

without splashing; and, further, to prevent the entrance of chips of wood, gravel, or other impurities which are likely to stop or otherwise obstruct the outlets. The first point is met either by having the supply launders expanded fan-like and furnished with dividing ledges, *b*, or by the interposition of small troughs, the sides of which nearest the box to be supplied are perforated near the bottom by equidistant small holes. The cleaning of the material, previous to its entering the first box, is generally effected by the main supply launder being made a little wider near the point of entrance, and the insertion at this place of a fine wire sieve across the launder, and somewhat inclined against the stream. This sieve must be occasionally looked after to remove any impurities collected in front; and this, in fact, is the chief attention the whole apparatus requires, for otherwise it needs hardly any supervision. If once in proper working order its action is constant and uniform, provided the material introduced does not change in amount and quality; and it has this further advantage, as compared to the slime-labyrinths, that the classified stuff can from the outlets be directly conveyed in small launders to the concentrating machines for treatment without any previous preparation.

One point, however, against the apparatus is, that having to be placed between the reduction mill and the concentrating machines, a great fall of ground is required to permit the direct introduction of the material, and also to allow sufficient fall for the tailings; and thus, where local circumstances are unfavourable, it has to be erected at a higher level. This necessitates the use of dipper wheels or other suitable appliance to lift the stuff.

The action of the different boxes on certain slimes, with regard to the percentage of fluid matter and the quantity and character of the solid contents respectively separated, has (according to Mr. Hunt) been ascertained by experiment as follows:—

The small box separated 38 to 40 per cent.; which contained per cubic foot 16 to 18 lbs. coarse sand.

The second box separated 20 to 22 per cent.; which contained per cubic foot 13 to 14 lbs. fine sand

The third box separated 18 to 20 per cent.; which contained per cubic foot 15 to 16 lbs. coarse slime.

The largest box separated 10 to 12 per cent.; which contained per cubic foot 10 to 12 lbs. fine slime.

Triangular Double Troughs.—Another method of classification, effected by means of double triangular troughs—the "Spitzlutten" apparatus—is based upon the principle that if material composed of particles different in size and density is exposed to a rising stream of water, the velocity of this stream may be so regulated that particles of certain size and character sink, and may be conveyed to concentrators, whilst the remainder is carried upwards. Consequently, by repeating this operation a certain number of times with a gradually decreased velocity of the rising stream each time, the material can thereby be separated into as many different classes of grains.

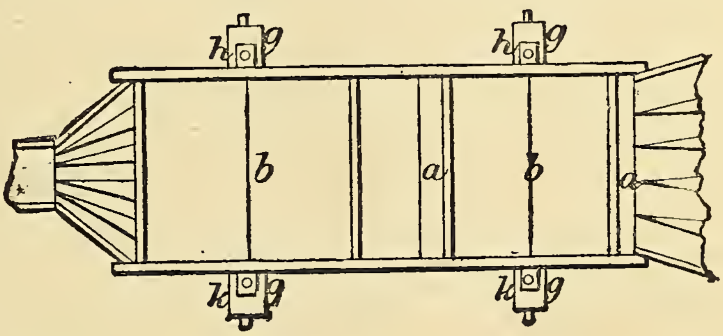


FIG. 85.—TRIANGULAR DOUBLE TROUGH. Plan.

The troughs (Fig. 86) by which this action is produced, are constructed as follows. Within a triangular trough, *a*, of certain length and width, with two opposite sides vertical and two inclined at angles of 60°, is a similar small one, *b*, having the vertical sides in common with the larger trough, but its inclined sides fixed at certain equal distances from and parallel to those of the latter. There is thus an open V-like space, *c*, left between the inclined sides

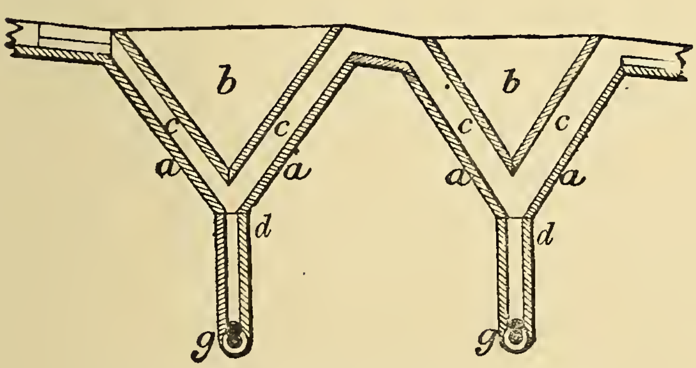


FIG. 86.—TRIANGULAR DOUBLE TROUGH.
Longitudinal Section.

of the two troughs, representing, as it were, a rectangular pipe sharply bent in the centre; and it is through this that the stream of material has to pass, *i.e.*, to fall and rise. The velocity of the stream depends on the size of the space, and also the size of the particles that will rise or sink in it. The cross section and respective velocity stand in inverse relation to each other, and their determination for each double trough of a complete apparatus is a matter of mathematical calculation, in which the size of the largest particles and the specific weight of the material to be classified form the main figures.

For ores which are crushed so fine that the largest grains are not more than 0·6 millimetre in diameter, the most satisfactory classification into four different kinds of grains is arrived at by a series of four double troughs, with the velocity of the stream decreasing from the first to the succeeding troughs, in the progression of 2·3, 0·94, 0·37, 0·15 in. per second, and if the width of the channel for the first trough is 1·1 in., and its length 2 ft., the dimensions of that of the second trough follow as 2·75 in. is to 2 ft. And as it is not advisable to increase the width of the channels beyond 3 in., the channels of the third and fourth troughs are each 3 in. wide, and respectively about 54.5 in. and 135 in. long. The mean depth of the channels, measured from the line of inflow of the material to the lowest part of the inside trough, is for the two smaller double troughs about 3 ft., for the two larger ones from 4 ft. to 6 ft.

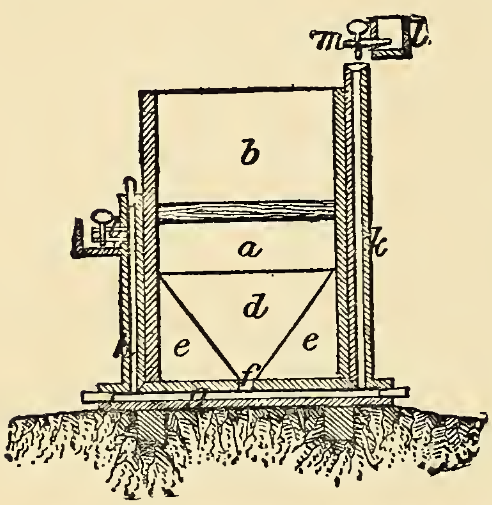


FIG. 87.—TRIANGULAR DOUBLE TROUGH. Cross Section.

In order to carry off the coarse particles that sink in the channels, the inclined sides of the outside troughs do not meet below, but are continued downward, forming a long and narrow pyramidal opening, *d*, about 1½ in. wide at the top. The

short sides, *e e*, slope inward at an angle of not less than 50°, contracting the opening to a small hole, *f*, of about 1 in. square at bottom, through which the material is discharged into a horizontal pipe, *g*, cross section, that extends both ways a small distance beyond the sides of the apparatus, and is connected at the ends with vertical 1-in. pipes. One of these, *h*, serves for the outlet of the classified material, and is carried up to within 36 in. to 21 in. of the water level in the channel, *c*, according to the degree of fineness of the particles that have to pass through it (the same as in the pyramidal boxes). At the top it is supplied with a tap for the regulation of the outflow. The other pipe, *k*, cross section, conveys a supply of clear water furnished from a launder, *l*, supplied with a tap, *m*, and as the water in the pipe stands 6 in. to 8 in. above the water level in the trough a small uniform pressure is produced, causing a forced influx of water at the point *f*, which is essential for good classification. This water opposing itself to the downward current charged with sediment in the pyramidal channel, *d*, prevents all but the coarser particles and pure water passing into the pipe, *h*, and thus only grains of the desired size are carried to the outlet, *i*. With regard to the relative positions of the different double troughs of the series, they are fixed exactly horizontal, and sufficiently below each other to prevent any settlement of material in the communication launders, which are necessarily very broad.

Other particulars regarding proper working, &c., are the same as those given for the pyramidal boxes. According to experience (says Mr. Hunt) a series of four of these double troughs classifies as well, and for the two coarser kinds even better, and cleaner, than a set of four pyramidal boxes, though for the fine slimes the latter are generally preferred, as they effect the desired settlement of the stuff more completely. A complete apparatus of troughs requires also less fall and space than one of pyramidal boxes, and is more easily regulated in cases of increased or diminished influx of material. As regards the results of classification by the different troughs of the series, they are approximately stated as follows: the first

or smallest trough separates from the material supplied about 30 per cent coarse sand; the second trough, about 25 per cent. fine sand; the third, 20 per cent. coarse slime; the fourth, 15 per cent. fine slime.

Conical Separator.—Mr. W. R. Raymond* gives an account of an apparatus of conical form which is illustrated below: the scale is $\frac{1}{25}$, and the upper cone is in section. A complete series is usually composed of five or six cones, arranged in succession, one below another, as shown. The construction is very simple, and they can be made of cast iron, so as to be very

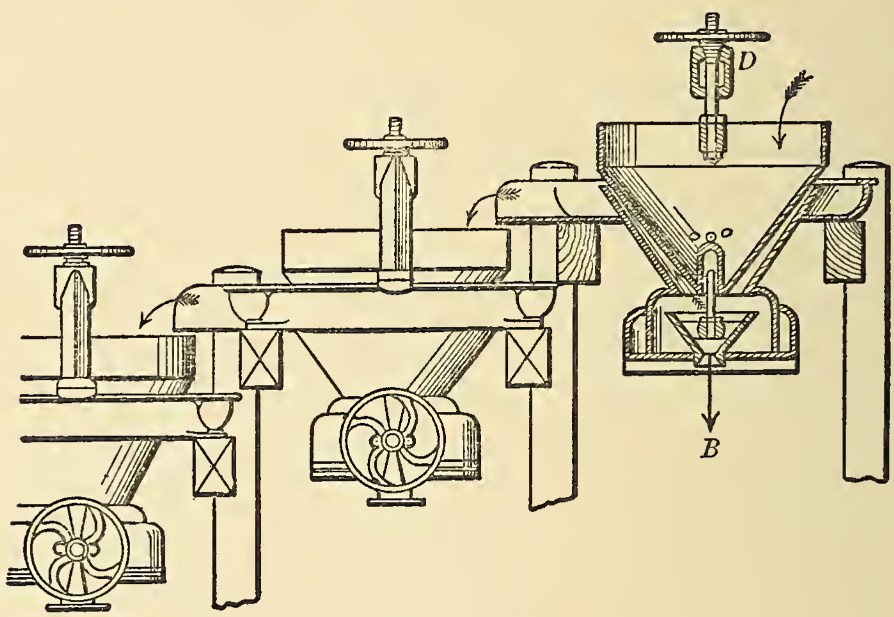


FIG. 88.—CONICAL SEPARATOR.

durable, and at the same time exact in form. Each part consists of two cones, one inserted in the other, so as to leave an annular space in which water flows upward from a reservoir or chamber at the lower or pointed end. The stuff to be concentrated is conveyed by a launder into the upper cone, and, passing through holes, encounters the upward current. The largest of the stuff so fed should not exceed three-quarters of a millimeter in diameter. The lighter portions are at once carried upward and over the upper edge of the inner cone, and fall with the escape water into an annular trough, by which they

* In his "Mineral Resources of the United States."

are conducted away to the next lower one, while the particles of sufficient weight to resist the current fall through it, and accumulate in a small inverted cone in the chamber below, from which they are allowed to drop by the small aperture at the apex in the direction indicated by the arrow, B. This orifice is controlled by a valve, and can be regulated at will, according to the rapidity of the accumulation. So, also by means of a screw, D, above the upper cone, the distance between the cones can be regulated according to the necessities of each case. The overflow from one cone is carried to the next, and so on in succession (Fig. 88).

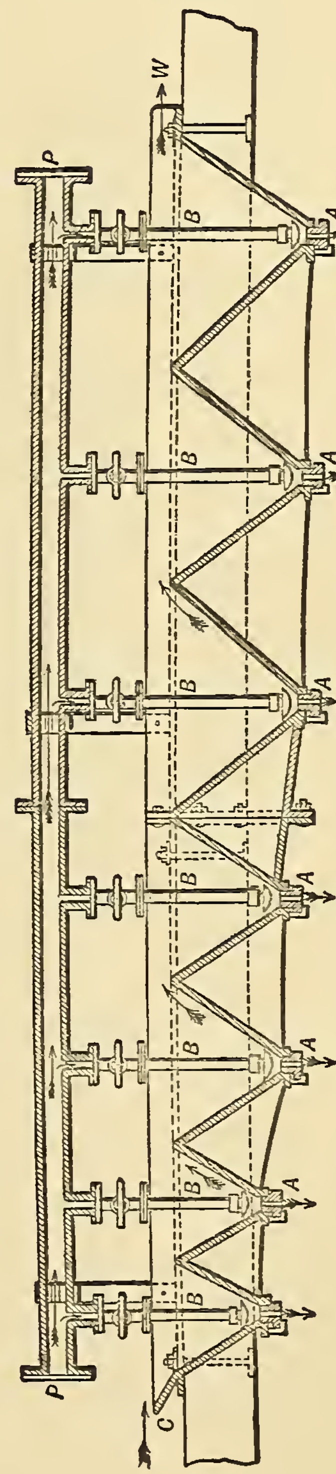


FIG. 89.—RITTINGER'S SEPARATING TUBS WITH ASCENDING CURRENTS.

Bilharz Spitzgerinne.—The appliance known by this name, Rittinger's separating tubs with ascending current, combines the principle of the flowing with that of the ascending current. The apparatus (Fig. 89) consists of a succession of deep trough-like depressions placed edge to edge, and gradually increasing in size and depth. But as the ends and sides are the highest, the series forms in reality but one vessel, the water covering all of the intermediate edges, and thus permitting a continuous flow from one end to the other. This will be seen from the inspection of the figure.

Seven compartments, B B, are shown, and the direction of the flow from C to W is indicated. The whole series is supported upon a frame at such a height that the attendant

can pass under it and reach the openings at the apex of the pyramidal tubs, at A A, where the concentrated stuff flows out. A supply pipe, P P, delivers clear water into each compartment trough, a branch pipe reaching nearly to the bottom. The stuff entering at C deposits the heaviest particles, and, aided by the ascending flow of water from the pipe, the lighter portions pass over into the next tub, and so on. The flow of water into each compartment must be carefully regulated. As the size of the compartments increase, the ascending current has less and less force, and finally only the very lightest and poorest portions are carried away.

The arrangement gives very satisfactory results. It requires from 120 to 150 quarts of water a minute, and will separate about a ton of battery pulp in each hour. It may be constructed either of wood or iron. The apparatus shown in the figure is made of iron.

CHAPTER XII.

CONCENTRATION AND ORE DRESSING (continued): CONCENTRATING.

Jigging Machines or Jigs—Coarse and Fine Sand Continuous Jig—Wimmer's Continuously Working Jig—Rittinger's Continuously Working Jig—Huet and Geyler's Self-Acting Jig—Argall's Jig—Collom's Jig—Concentrating on Buddles—The Convex Buddle—The Concave Buddle—The Percussion Table—The Rotating Table—The Embrey Concentrator—The Krom Dry Concentrator.

I shall now proceed to the description of apparatus for treating sized stuff.

Jigging Machines, or Jigs.—The idea of a "jig" was originally derived from the treatment by hand of ore on a sieve under water. By plunging the sieve down suddenly in the water, and allowing the particles to come again to rest upon it, a separation is effected, and if the stuff has been sized, and the operation has been sufficiently often repeated, the denser particles are found in strata under the less dense. If the mass on the sieve is then divided into horizontal layers, ore and gangue may be separated.* The first mechanical jig had contrivances for imparting motion to the sieve, but it was afterwards found that the same results could be obtained by using a submerged stationary sieve and imparting a vertical oscillating motion to the water. This is done either by means of pistons or elastic pliable plates, or rubber cloth, or some such material, placed in the sides of the box or on the top of a lower chamber full of water, and communicating with a box containing the sieve.

* This primitive method of concentration by hand-power I saw in practice at the La Union Mines, near Carthagena, in Spain, for the separation of zinc blende and galena.

An additional feature of recent mechanical jigs is the continuous discharge.

Piston jigs may be divided into two classes: those in which the piston is placed below the sieve, and those in which it is situated at the side or behind the sieve, in a box communicating freely below with the box containing the sieve.

The piston imparts a pulsating movement to the water, and if two bodies of equal volume and of distinct specific gravity be dropped at the same instant from one and the same height in a column of water, the one of greater weight will increasingly leave the other and arrive at the bottom first. The light and worthless particles of ore are removed by the flowing action of water in the continuous jiggers.

For the purpose of effecting a good separation of ore from its gangue it is necessary to give great attention to the length of plunger stroke, number of strokes per minute, and volume of water flowing into the jigger. Every distinct class of vein stuff will necessitate the observance of some special condition, so that no absolute directions can be given. The practice is to increase the speed, shorten the stroke, and lessen the thickness of bed with the decreasing size of grains to be treated. For coarse sand jiggers the speed may be from 60 to 75 strokes per minute, while the water required will be from 9 to 15 gallons per machine per minute. In fine sand jiggers the speed will vary from 90 to 150 strokes per minute, and the water from 10 to 15 gallons per minute. In slime jiggers from 150 to 300 strokes per minute are made. The length of stroke varies from ¼ in. to 2½ in.; the coarser the ore the longer the stroke.

The Continuous Jigger (Fig. 90) is successfully applied to the enrichment of both coarse and fine sands. Instead of separate sieves, placed one below another, one long and slightly inclined sieve is employed. This sieve, supported on a wooden grid, is covered with a second grid of similar construction, in the compartment in which the bed is lodged.

Underneath the sieve the ore and dredge receptacles are placed, and at the end of the hutch is a waste box with a

launder for the escape of the water. The bottom edge of this

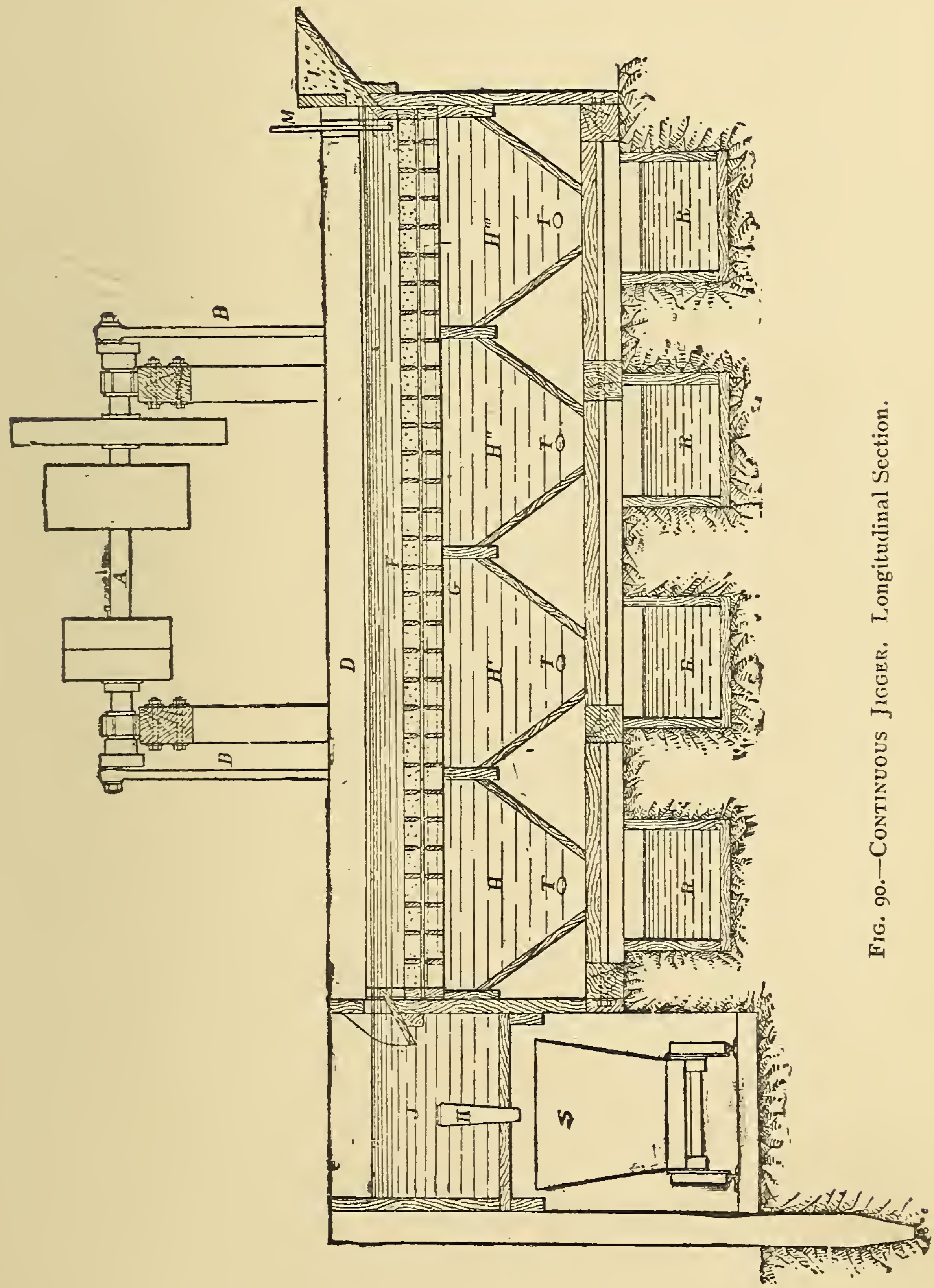


Fig. 90.—Continuous Jigger. Longitudinal Section.

launder is from 2 to 3 inches above the level of the sand supposed to be in the jigger. The jigging, or separation of the

stuff, is consequently performed under water, and any fine slime which the stuff may contain floats and leaves the grains of which the stuff is composed free to separate, fall, and arrange themselves according to their respective densities.

A jigger 20 ft. long, provided with a sieve 42 in. wide, will dispatch approximately—

Stuff composed of grains 3 to 5 millimetres diameter, 5 tons per hour.
" " " 2 " 3 " " 3 " "

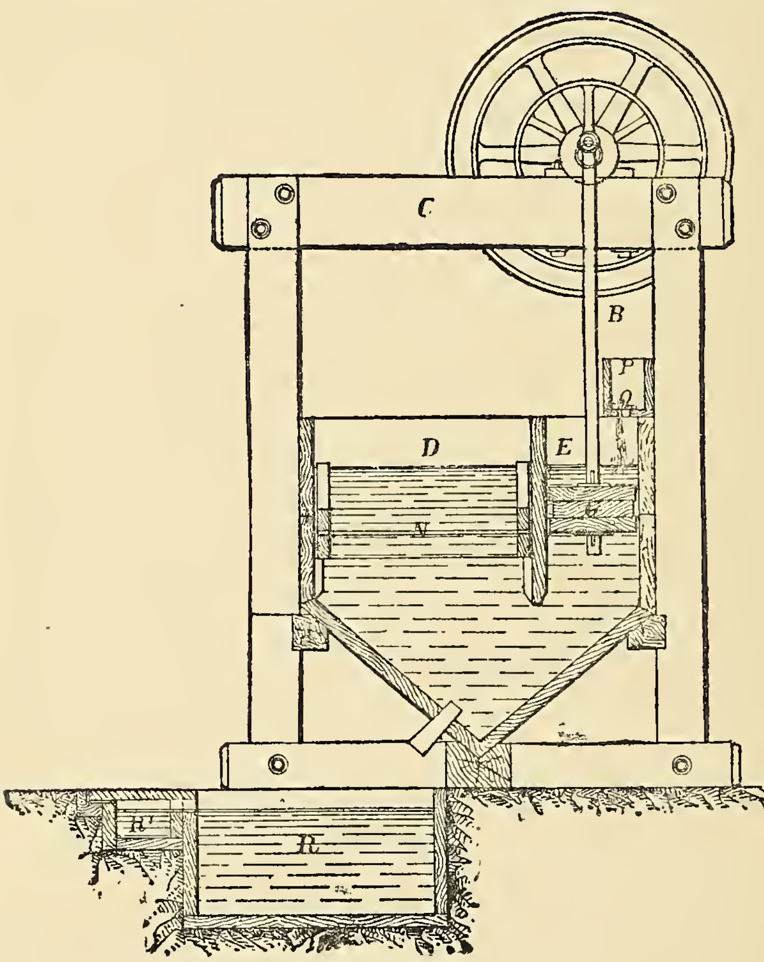


FIG. 91.—CONTINUOUS JIGGER. Transverse Section.

A, driving gear; B, adjustable eccentric piston rods; C, framework; D, sieve compartment; E, piston compartment; F, ragging frame; G, grid frame supporting sieve; H H′ H″ H‴, ore boxes; J, waste box; K, plug for discharging waste box; L, feed hopper; M, slide for adjusting rate of feed; N, wirework bottom; O, hole in launder; P, partly covered with a loose slip of iron for admitting water to jigger; R, ore boxes; S, waste wagon; T, plug holes for discharging ore from boxes H H′ H″ H‴; R, launder for taking off the overflow of water from the ore boxes, R.

Wimmer's Continuously Working Jig is a very simple form of piston jig, as shown in Fig. 92. It is made of wood,

with a piston or plunger, P, at one side, which, on being forced downward upon the water in the box, causes an upward flow through the grate in the direction *b* to *c*. The peculiarity of this construction is a valve in the centre of the sieve through which the concentrated stuff is delivered as it accumulates, while the refuse passes off over the partition in front. But it was found that the downward current of water when this valve was opened was sufficient to carry down some of the waste stuff from the top, and it became necessary to devise some means of preventing this flow. This was effected by covering the outlet with a conical tube, *d*, supported from a bar of wood above and reaching down through the layer of poor stuff so low that only the heavy and richer portions resting directly upon or near the sieve can pass downward into the discharge pipe, *b f*. This pipe is alternately opened and closed at the top of an iron stopper placed at the end of a vertical rod, the upper part of which slides through a supporting ring, *g*. By means of an arm, *i*, supported on a pivot at *k*, the stopper is alternately raised and lowered as the piston, P, rises and falls. The opening in the discharge pipe is thus opened when the piston descends, and is closed when it ascends. It has been found in practice, however, that this arrangement for opening and closing the discharge pipe does not give satisfactory results.

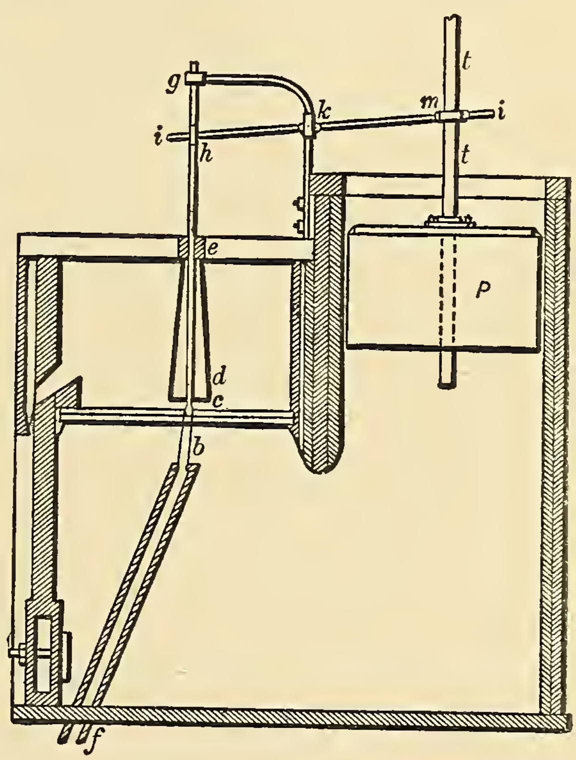


FIG. 92.—WIMMER'S CONTINUOUSLY-WORKING JIG.

Rittinger's Self-Acting Jig.—This jig (which is shown in Fig. 93) is characterised by the inclination of the grates and the lowness of the front partition, over which the poor and

lighter stuff falls continuously, and with very little water, while the heavier and richer portions fall through the opening or slit, *o*, at the base of the partition. This partition is the segment of a cylinder, and is supported upon the lever or arm, *d*, so as to be movable backward and forward in such manner that the opening, *o*, may be increased or diminished at will. The heavy stuff passing through the opening falls into the box, K, from which it is removed as required. The inclination of the grate in this machine is from 5° to 8°. It is fed through the hopper, B, which plunges below the surface of the stuff accumulated on the grate. The loss of water which occurs at each stroke of the piston is replaced from a reservoir, W, at the back of the apparatus.

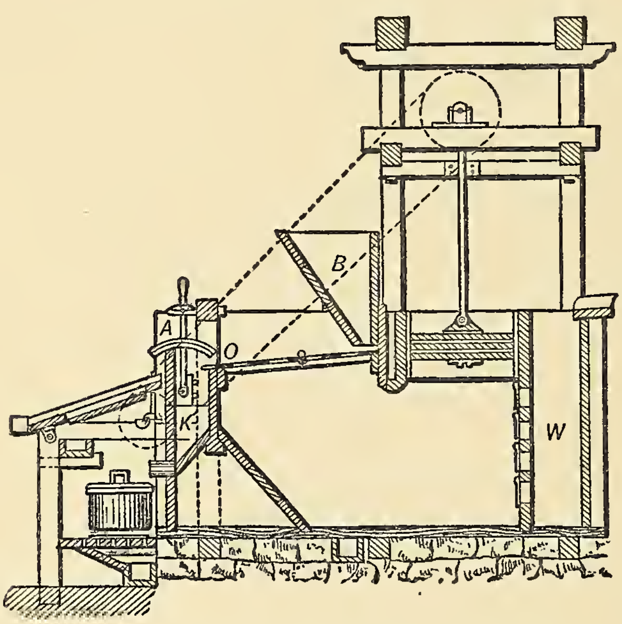


FIG. 93.—RITTINGER'S SELF-ACTING JIG.

Huet and Geyler's Self-acting Jig.*—This appliance (Fig. 94) is constructed of cast iron, and is very compact. Most self-acting jigs require a large quantity of water, and this in many places is a great objection to their use; but this jig is designed to work with but little loss of water, and at the same time, by the aid of an automatic scraper, to increase the product.

The tube is shaped like the letter U, and is divided into two compartments, one for the piston and the other for the working grate. Water enters through the valve, A, at the side, and the fine stuff or slime which falls through the sieve settles upon the bottom, and is discharged through an opening, B, controlled by a lever reaching out to the front of the apparatus. The piston is operated by means of a shaft and crank, which works in an

* Described by R. W. Raymond in his "Mineral Resources of the United States."

inclined slide, C, connected with a lever carrying the piston, so as to give a rapid descending stroke with a period of rest at the bottom, and then a slow upward movement, thus giving the most favourable conditions for the rapid and perfect separation of the stuff upon the grate.

The motion of the piston may be varied at will, in order to secure the best flow or motion of the water for different grades of ore. This adjustment is effected by shifting the position of the head of the piston along the lever or arm, and by this means increasing or diminishing the amplitude of its motion.

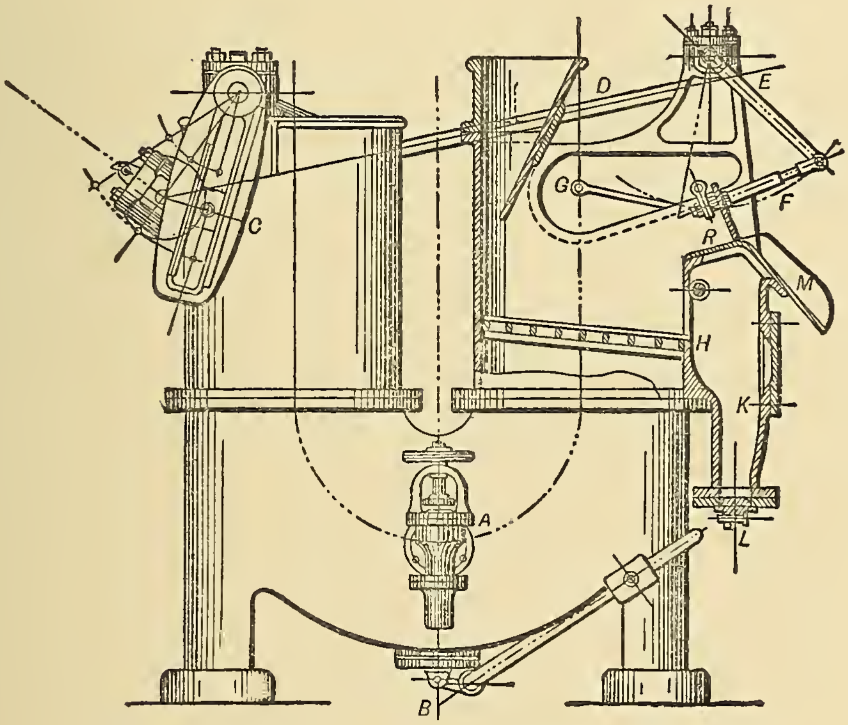


FIG. 94.—HUET AND GEYLER'S SELF-ACTING JIG.

The construction of this slide is shown in Fig. 95. By turning the fixed screws, *s s*, the head of the piston may be moved forward or backward.

The machine is provided with a scraper, R, actuated by the long rod, D, which is attached to an eccentric on the main shaft, and moves the levers E and F, giving to the scraper a forward and backward motion over the top of the stuff upon the grate, and throwing out a portion of it at each movement. The path of the scraper is determined by the guides, G, attached

to each side of the tub. It can be varied by means of screws upon the lever or arm, F. In passing backward, the roller or projection on the scraper which follows the guides rises upon the movable inclined plane, G, and on its return passes below this plane, following the double-dotted line in the figure. The poor stuff from the top, which is constantly thrown forward and off by this scraper, falls over the front of the tub at R, along the shute, M. The grate is inclined, as in the machine of Rittinger, and the opening for the escape of the heavier and rich portion is similarly placed at the foot of the incline, and just below the bridge over which the poor stuff is scraped. The opening is shown at H. It is closed by a valve which extends along the whole front edge of the sieve, and can be opened and closed at pleasure by a lever. The stuff passing through this valve falls into a receptacle, K, from which it may be removed at pleasure through the opening, L. The scraper is so made of perforated sheet iron that it does not throw the water out together with the waste.

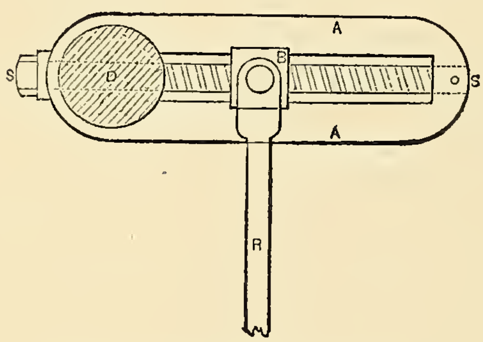


FIG. 95.—SLIDE TO HUET AND GEYLER'S SELF-ACTING JIG.

Argall's Jigger.—The piston in this jigger, placed between two hutches, is in free communication with two sieves, S S (Fig. 96). Two sizes of ore may be jigged at the same time on each side of the piston. P, piston; S S, sieves; D D, discharge openings. The hutch is built and supported by a frame of wood, to which it is securely bolted. This frame extends above the hutch, and carries the eccentric shaft and gear.

This machine, with a "four-hole" sieve and a speed of a hundred and fifty revolutions, will jig from 5 to 6 tons of coarse-grained stuff per hour. The pistons are hung to eccentrics. Gear is fixed to carry off the dredge as it accumulates, while similar gear is attached to keep the bed at a constant level. The waste passes from the end of the machine into a waggon,

while the concentrated ore is continuously discharged from the hutch. Figure 97 shows the method of constructing these machines in wood, and of mounting the driving gear on a wooden frame. The machine includes three rectangular-shaped

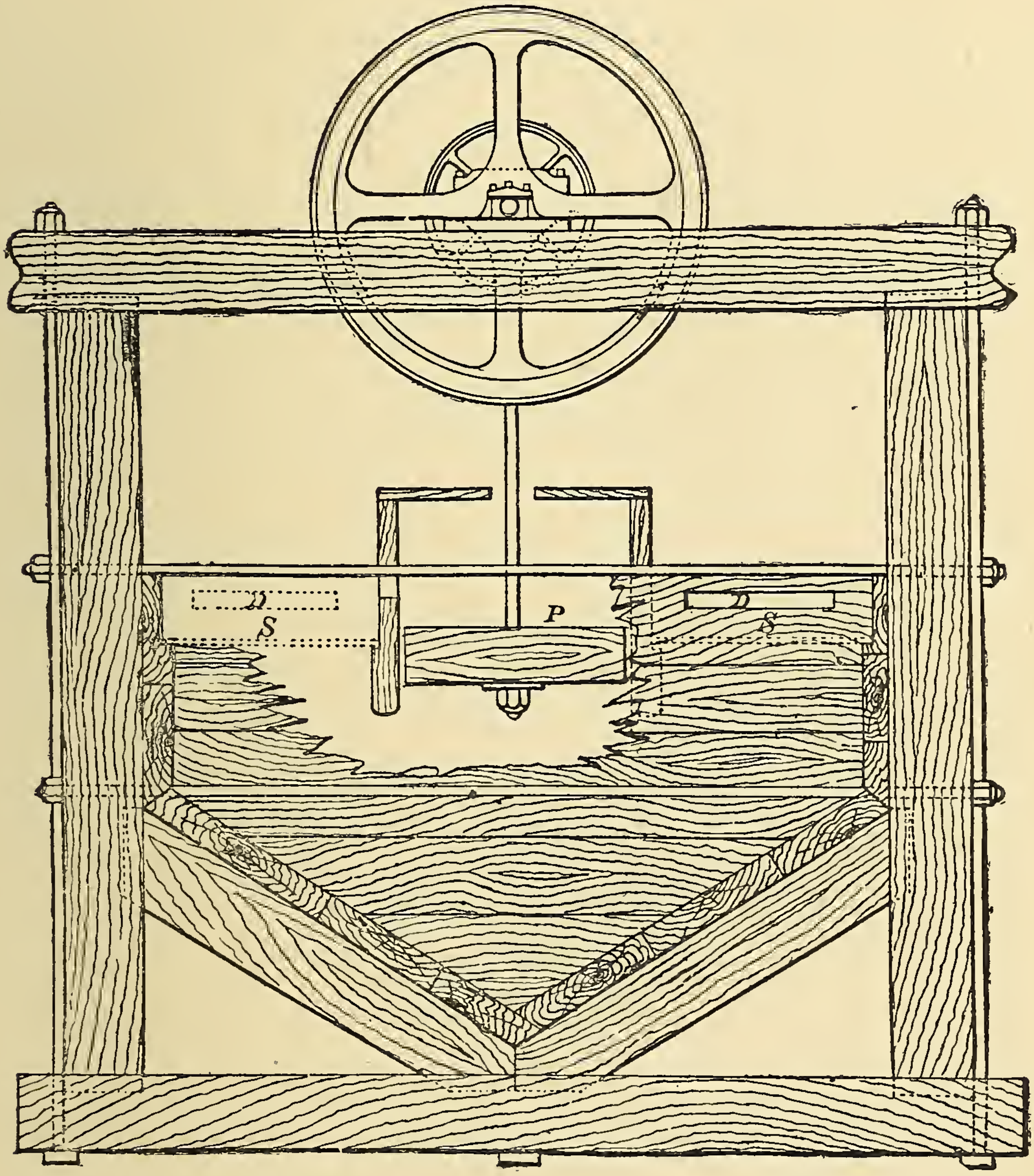


FIG. 96.—ARGALL'S JIGGER. Section.

pistons and three cascade bottoms. Shifting eccentrics for varying the length of stroke in the pistons and a fly-wheel to secure regularity of motion are employed. The concentrated ore is drawn from the ore chambers on lifting the slides by means of the respective hand levers.

Collom's Jigger.—This is, in fact, a combination of two machines, which—for convenience put together as one—are

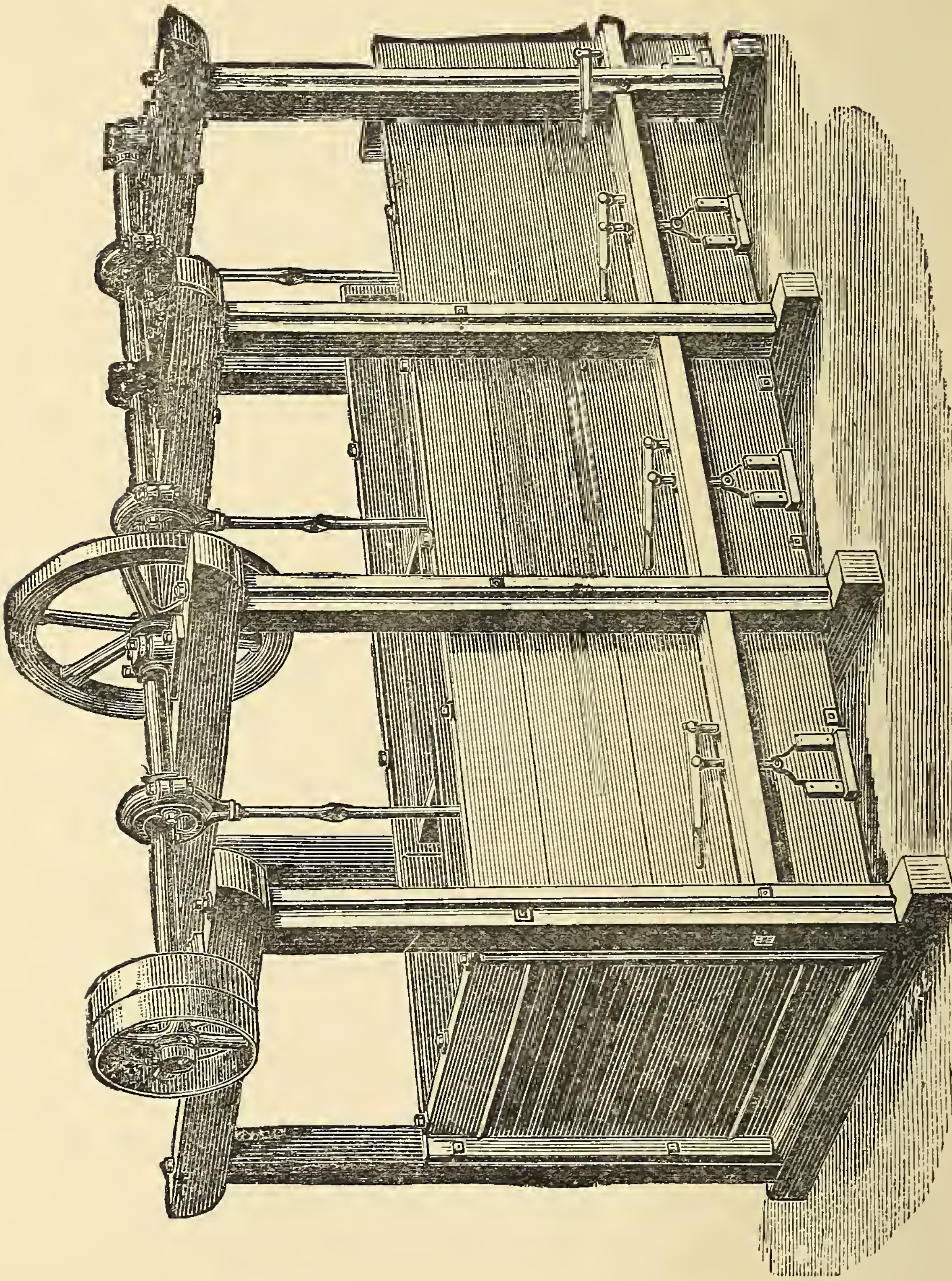
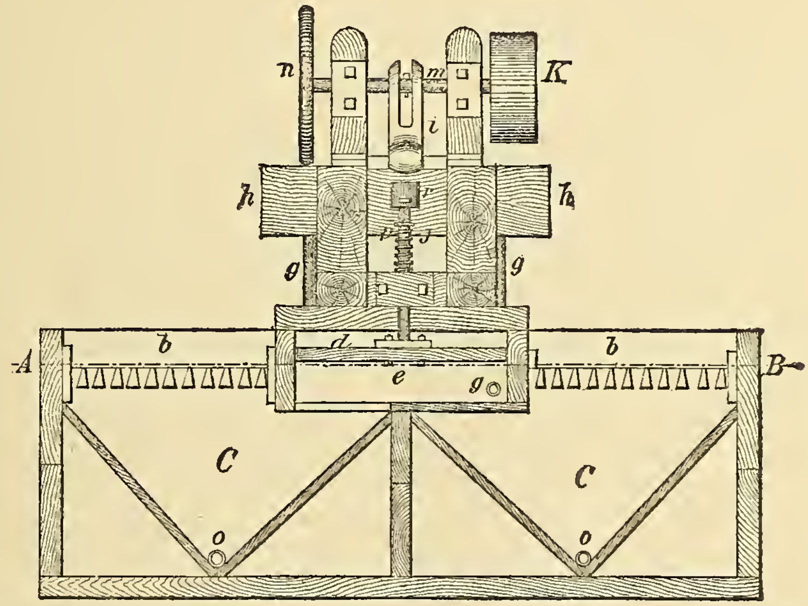


Fig. 97.—Argall's Jig.

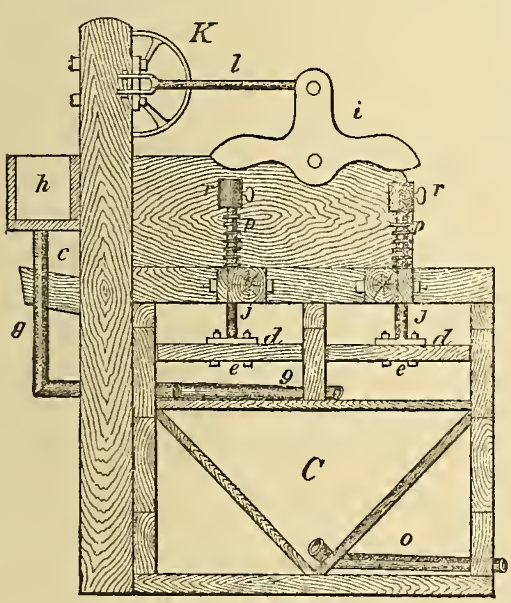
quite independent of each other in their operation. A double machine consists of a box or tank about 7 ft. long and between

3 and 4 ft. wide, divided by a middle partition (as shown in Fig. 98) into two parts. Each of these parts is fitted on the inside with inclined partitions, sloping from the four sides towards the centre of the box, as shown in Figs. 98 and 99, and thus forming two cisterns, C, above each of which is placed a sieve, *b*. The sieve frame may be furnished with a wire cloth sieve of any desired degree of fineness, according to the character of the ore to be dressed.



Scale $\frac{1}{32}$.

FIG. 98.—COLLOM'S JIGGER. Longitudinal Section.



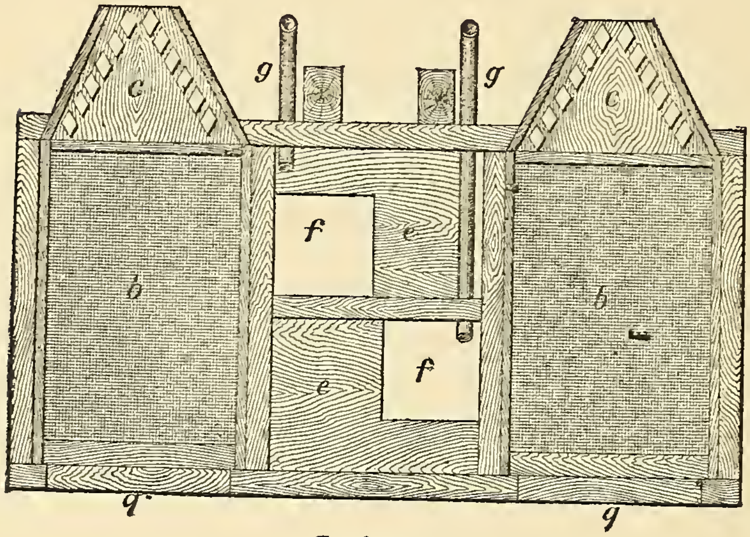
Scale $\frac{1}{32}$.

FIG. 99.—COLLOM'S JIGGER. Transverse Section.

Between the two sieves are the piston or plunger compartments, *e*, separated from each other, and each connecting by an aperture, *f*, with one of the cisterns, C. Each aperture, *f*, affords communication with the cistern nearest to it, but without any connection with the other cistern. The plungers, *d*, move up and down in the compartments, *e*, being forced rapidly downward by rockers, *i*, and lifted again by the action of springs, *p*. The rockers are set in motion by pulleys, K, with which they are connected by eccentric rods, *l*.

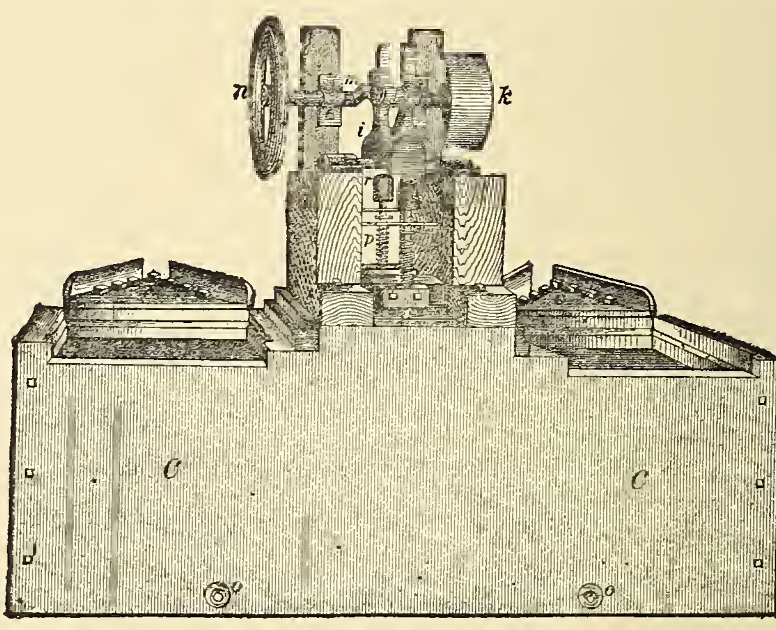
The cisterns and plunger compartments are supplied

with water by pipes, g, and when the outlets, o, are closed, the machines are filled with water, the overflow being at q, in front of the sieves. The movements of the plungers, therefore, will follow each other in rapid succession, produce an agitation of the water, which rises through the sieves with a constantly throbbing motion. The crushed ores, consisting of heavy mineral



Scale $\frac{1}{32}$.

FIG. 100.—COLLOM'S JIGGER. Horizontal Section.



Scale $\frac{1}{32}$.

FIG. 101.—COLLOM'S JIGGER. Perspective View.

References to Collom's Jigger.

C, Cistern or hutch.	*g*, Water pipes.	*m*, Crank shaft.
b, Sieves.	*h*, Water trough.	*n*, Balance wheel.
c, Buttoned head.	*i*, Rockers.	*o*, Outlet pipe for ore.
d, Plungers.	*j*, Plunger stems.	*p*, Springs.
e, Plunger case.	*k*, Pulley.	*q*, Overflow for waste sand.
f, Apertures.	*l*, Connecting rod.	*r*, Adjustable thimbles.

and gangue, are brought upon the sieves, b, by a stream of

water that enters through the distributing boards, *c*, and being subjected to the agitation caused by the plungers, *d*, are held in a state of partial suspension, during which the heavier metallic particles sink, while the earthy matters rise to the top and are carried off by the water at the overflow, *q*. That portion of the metallic substance which is fine enough to pass the meshes of the sieve falls through into the hutch or cistern, C, and may be withdrawn thence at stated intervals by the outlet pipe, *o*, while the coarser part remains upon the sieve and is cleaned up from time to time, leaving a stratum on the sieve for continued operations. The thimbles, *r*, on the plunger rods, *p*, serve to adjust the length of the stroke. The action of this machine is excellent.

There are establishments where four of these machines are used, or eight sieves. Two of the double machines, containing four sieves, stand on a raised floor sufficiently elevated above the other two that the material delivered from the outlet pipes, *o*, of the first may flow to the sieves of the second. One of the upper machines and one of the lower, immediately in front of it, are furnished with No. 6 sieves for washing the coarser material, while the other two, upper and lower, are furnished with No. 10 sieves for the finer stuff. The ore that enters upon the upper sieve is therefore rewashed on the lower sieves, in order to insure a more effective separation. The overflow of the two upper sieves, of either degree of fineness, that is, the material discharged at *q*, is washed again upon one of the lower sieves of the same degree of fineness, the overflow from that sieve being worthless gangue, while that which passes through the sieve is second quality ore, such as blende and copper mixed. The stuff that passes through the two upper sieves of either degree of fineness, is delivered from the outlet pipes, *o*, and comes upon the remaining sieve of corresponding degree of fineness, the material which passes through that sieve being of first quality, while the overflow at *q* is of second quality.

By this arrangement there are three products obtained: first, the pure galena, which is almost entirely free from other mineral; second, the zinc blende and grey copper, mixed with

heavy spar and quartz, almost free from galena; and, third, the gangue, which is very clean and free from valuable mineral. This result is reported by Mr. C. King as having been obtained at a mine in Colorado.

The eight sieves, or four double machines, are capable of treating 20 to 30 tons of ore per day; and as the stuff is all washed twice, the capacity of each double machine, for a single washing, is from 10 to 15 tons per day.

Concentration on Buddles.—By "buddles" are understood machines for the concentration of slimes and fine sediments on a circular bottom. They were introduced into California by the Cornish miners, and though they may be considered old-fashioned they are very reliable. The buddle may be arranged with the feed at the centre, and the discharge on the circumference, in which case the surface of the buddle will form the frustrum of a cone; or the feed may be at the circumference and the discharge in the centre, in which case the surface of the buddle will form an inverted frustrum of a cone.

The Convex Buddle.—A buddle of the first kind is shown in the accompanying illustration (Fig. 102). The slime—coming through the trough, *r*—enters the funnel-shaped receptacle, *c*, and through suitable openings passes over the conical surface, *h*, and thence up on the bed of the buddle, *a a*. The vertical central shaft, *s*, receives, by means of bevel gear, *t*, a rotary motion from the shaft, *s*. The arms, *d*, and the receptacle, *c*, are in connection with the revolving shaft, *s*; and in the arms, *d*, are attached the rollers, *n n* and *n' n'*, which are provided with cranks and catches. To these are attached the brushes, *f f* and *f' f'*, which serve to smooth and consolidate the ore on the buddle.

The circumference of the buddle is enclosed by the wooden partition, *a'*, 12 in. high, which is provided with round holes at different heights. These holes are successively closed with wooden stoppers as the ore rises on the surface of the buddle. Instead of brushes attached to the pieces *f f* and *f' f'*, canvas

cloths are frequently used with good effect. The diameter of the outer circle of the buddle shown in the illustration is

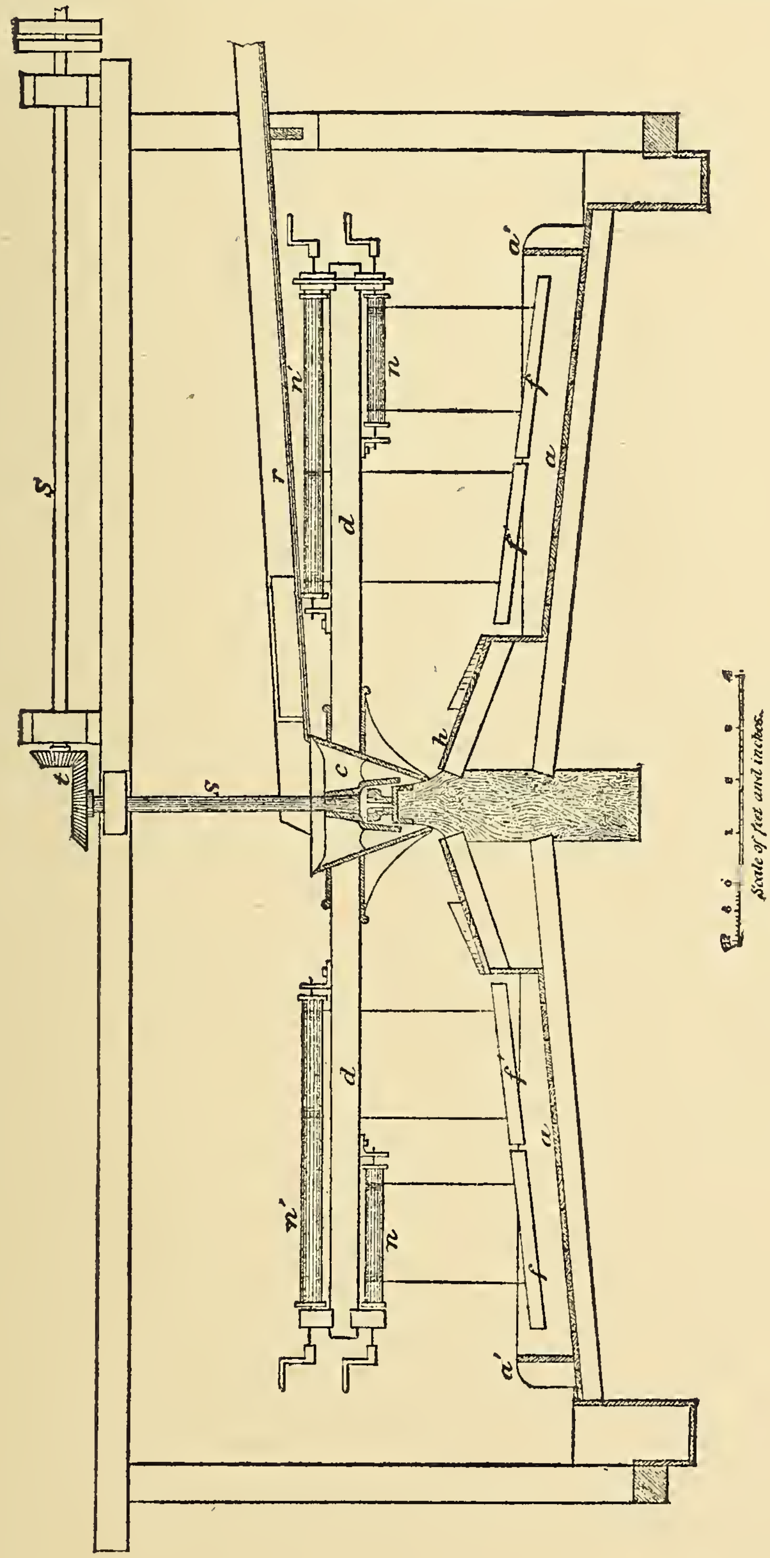


Fig. 102.—The Convex Buddle.

20 ft., and that of the conical table in the centre 6 ft., so that the length of the conical surface over which flows the material to be dressed is 7 ft.

The central shaft should make from ten to twelve revolutions per minute, and requires but very little power to drive it. Rittinger estimates the force required at a-twentieth of one-horse power. The rollers *n n* and *n' n'* regulate the position of the wooden bars, *f f* and *f' f'*, which carry the brushes or cloths. The water on the buddle should carry from 40 to 60 lbs. of fine ore to the cubic foot, and from 2 to 3 cubic ft. of it should be allowed to flow on the buddle per minute. The inclination of the surface of the buddle should be such that the outer edge is from 4 to 8 in. lower than the inner circle, where it is fed. The inclination varies with the fineness of the ore treated, it being, of course, greater for coarse stuff than for fine. The time necessary for filling a buddle of the given dimensions varies from two to three hours, according to the fineness of the stuff.

Buddles are well calculated for washing equal falling grains, except when they consist of very fine or light slimes. These remain too loosely on the surface, in which the water soon forms furrows or channels. There is, of course, no separation accomplished by buddling when the water flows in thick streams. Slimes of this consistency can be better treated on plane tables.

Concerning the relative merits of the convex and concave forms, the following opinion of Goetschmann may be cited: "The convex buddle has several disadvantages, chief among which is the retardation of the flow of the slimes, as they spread out in passing from centre to circumference, by which the current gradually loses the power of carrying away the worthless portions of suspended material. The device of feeding farther from the centre only gives a narrower ring, and hence a shorter distance for the flow. Moreover, the inclination cannot be changed to suit the nature of the material, yet is apt to change itself unsuitably, becoming steeper by the accumulation of headings around the centre. Finally, a very serious drawback is the absence of any means for recovering during the same operation any valuable portions which have been once carried too far down by the stream. For a com-

plete ultimate or repeated separation the ordinary convex buddle is therefore unsuited. Its best function is the preparation of material for the percussion table.

"The concave buddle was devised to obviate the evils above referred to, connected with the outward flow. In this apparatus it will be seen the working surface becomes smaller as the quantity of suspended material decreases in the water, and at the same time the force of the current increases. Theoretically it is therefore the better machine than the convex buddle, and this much is generally confirmed in practice, though the superiority claimed for it over the percussion table and some other machines is disputed. It is not likely to be suitable for complete separation."

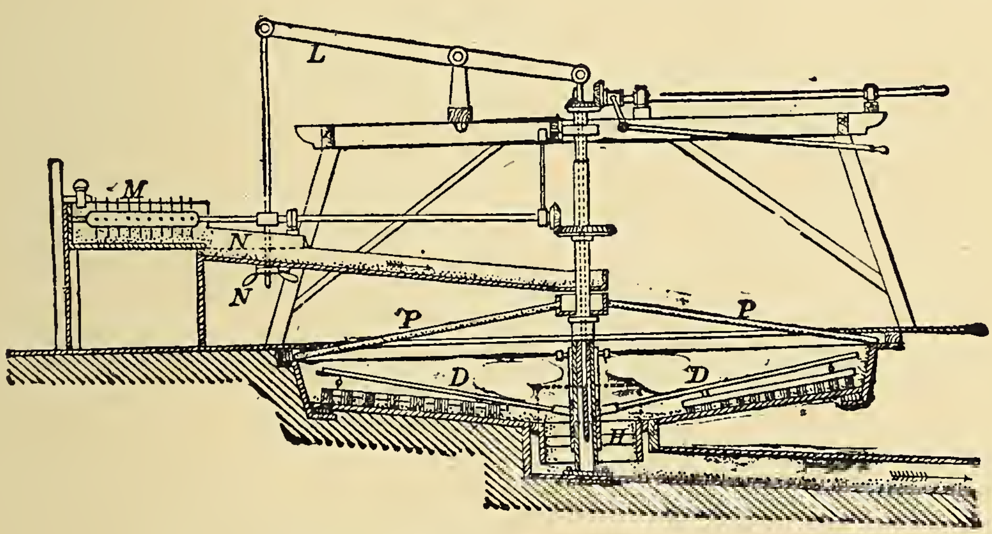


FIG. 103.—BORLASE'S CONCAVE BUDDLE. Section.

Concave Buddle.—The heads from several round buddles are usually thrown into a trough or launder, into which a stream of clear water flows of sufficient volume to convey the stuff to concave buddles. Fig. 103 shows Borlase's concave buddle in elevation and plan with a mechanical arrangement for adjusting the level of the central outflow by using a ring, H, that slides upon the centre vertical shaft.

By this means the height of the outflow is adjusted more gradually and uniformly than by the plugged holes in the ordinary buddles, and there is less liability to waste by gutter-

ing; the sliding ring, H, is raised by hand by the rod and lever, L, provided with adjusting fly nuts, N, and the arms of the sweeps, D, being supported upon the rising ring, are kept at the proper height by the same adjustment; the stuff is introduced into the box, M, and prepared for the buddle by means of the revolving agitator; it then passes through a perforated plate, N, down the launder to a central box, from whence it is distributed to a circular ledge of the buddle by six revolving spouts, P, from which it flows uniformly over the conical floor, falling at

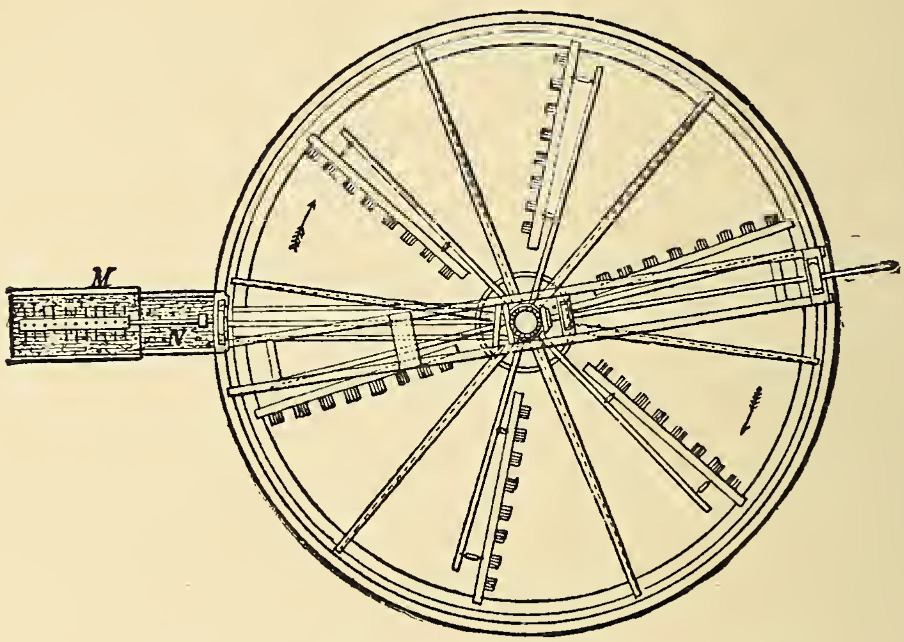


FIG. 104.—BORLASE'S CONCAVE BUDDLE. Plan.

a slope of about 1 in 12, towards the centre, H. The proportion of the ore is deposited round the circumferance of the floor, while the slime and waste flow over the top of the rising ring into the well, H.

The Percussion Table (Fig. 105).—Another plan for smoothing and consolidating the surface of ore deposited on a plane surface has been employed in the machine commonly known as the percussion table. The accompanying drawing of a percussion table of the most approved construction shows the form and the mode of feeding and imparting motion to these machines.

The hardening and evening of the ore bed is accomplished

in a percussion table entirely by mechanical means, no manual labour being required to work it, except for dividing the charge

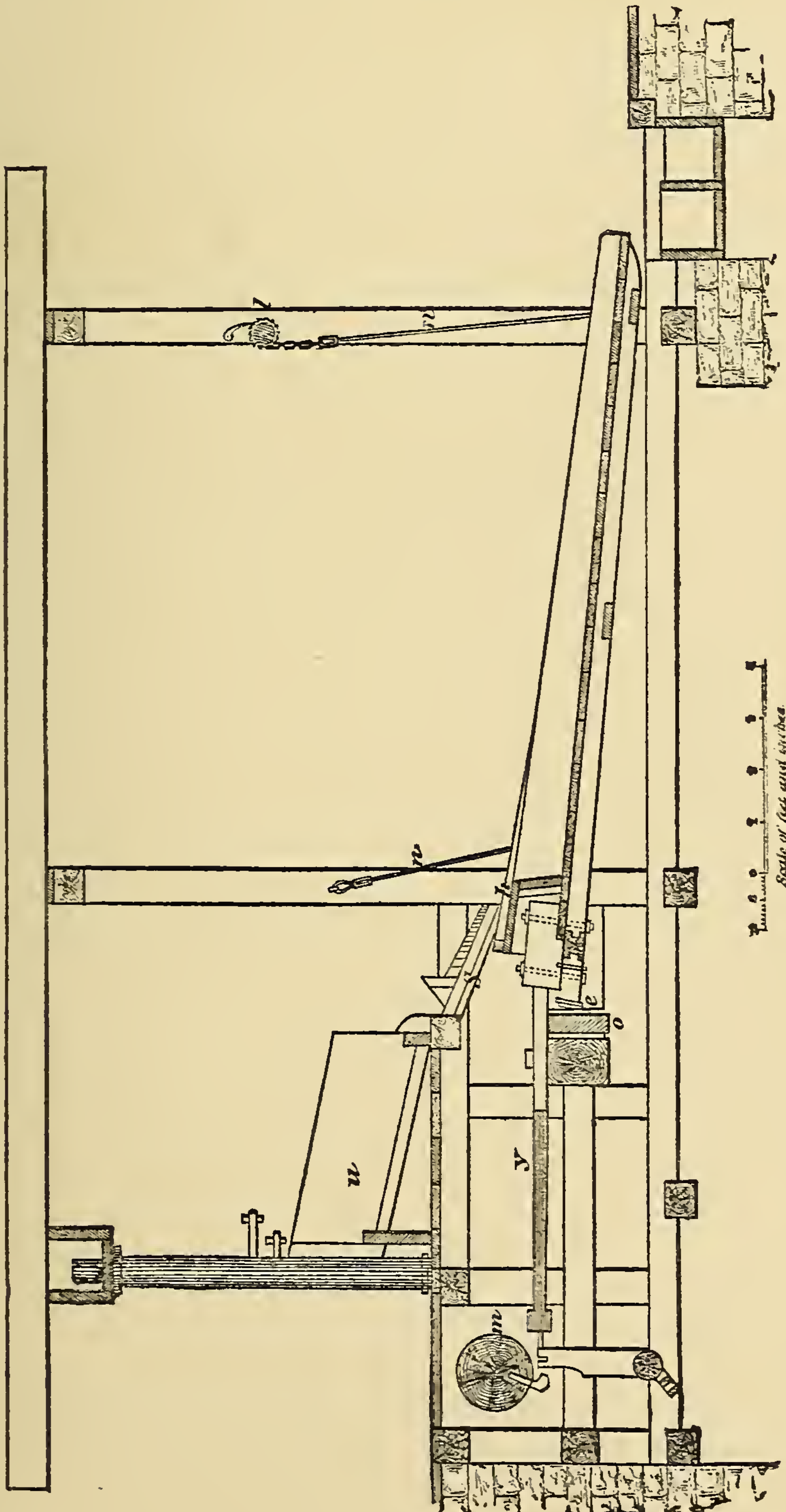


Fig. 105.—The Percussion Table.

into sections at right angles to the longitudinal axis of the table, removing the old and preparing the table for a new charge.

In the percussion table the shock given to the table is imparted to the particles, so that even the finer particles which remain loose on the buddle, and which have to be stamped or pressed down by hand on the plane table, are thoroughly shaken together and consolidated by mechanical means on the percussion table.

The blow imparted to the percussion table could be caused by striking the movable table with a movable weight, but it is ordinarily given by suspending the table, swinging it from its position of equilibrium and allowing its backward swing to be stopped by striking a stationary object. The blow may be imparted to the end or to the side of the table.

By reference to the drawing the operation of sizing equal falling grains on a percussion table will be easily understood. The ore is fed, in the first place, into the box, *u*, at the head of the table, and thoroughly mixed with water, which flows upon it through the stop cocks shown immediately above it. The water carrying the ore flows over the board, *s*, and is evenly distributed in the thin stream over its surface by means of a row of stout wooden pegs. In this condition the water flows upon the board, *h*, attached to the head of the percussion table, and from which it flows upon the table itself, which is usually made 12 feet long by 5 feet broad. Three longitudinal and three cross pieces enter into the construction of the frame for the table. The boards composing the floor are not tongued and grooved, but simply driven up close with a hammer and nailed fast, after a strip of lamp wick has been laid between them. The lumber need not be more than about half seasoned, as shrinkage is not to be feared when the table is wet, and the expansion of perfectly dry wood might be sufficient to twist and "buckle up" the floor.

Motion is imparted to the table from the cam shaft, *m*, by means of the rod, *y*. The table being hung by the rods, *n*, *n*, is swung at each revolution of the shaft, and on returning the head, *e*, strikes against the bumper, *o*, thereby giving a sudden jar to the table and its contents.

The swinging of the table, both out and back, must be slow

enough to allow the stream passing over it to take part in the motion, and flow on over the surface of the table without being materially retarded by the outward, or accelerated by the backward swing. The momentary stoppage of so shallow a stream would permit the lighter particles to deposit themselves in the ore layer at one time, and the accelerated current would carry heavy particles along with it at another time.

By means of a long lever and the roller, *l*, the lower end of the table can be raised or lowered, so as to give it the proper inclination and the current proper velocity, which must be determined practically according to the kind and size of the ore being dressed. It varies from 4 to 6 in. for slimes, and from 10 to 16 in. for coarse stuff in a table 12 ft. long. In dressing coarse stuff from 0·5 to 0·7 cubic foot of the watery mixture of ore is fed on the table per minute; and of slimes not more than from 0·1 to 0·14 cubic foot. In the former case the water should contain from 20 to 40 lbs. of ore per cubic foot, and in the latter not more than from 5 to 10 lbs.

A comparatively large number of small particles is required to make up the weight of one large particle. If the smaller ones are half the diameter of the larger ones, it takes four small ones to make one big one, and as the particles must be free to move in the current, it is evident that much more water is required for a given weight of fine ore than for the same weight of coarse ore.

When there is a fall of twelve inches in the length of the table, the velocity of the water current will be about a foot per second. The velocity of the motion of the table in a horizontal direction should be somewhat less than that of water.

It is evident from the drawing that the rods by which the table is hung vary considerably from the perpendicular. The cord of the arc through which the table moves is therefore inclined also, and the table partakes somewhat of a vertically oscillating motion.

When the length of the rods by which the table is suspended is 4 feet, the distance of the lower end from the perpendicular through the upper end should be 6 inches for dressing coarse

stuff and 10 inches for slimes. The horizontal movement of the table varies from 5 to 0·5 inches according to the coarseness of the material; and the vertical distance through which the table falls varies from 0·90 to 0.11 inch. In each case the larger number is for coarser stuff.

The action of the table depends somewhat on the elasticity of the bumper against which it strikes. If a block of rubber is placed on the face of the bumper the table will strike several blows for each time that it is pushed out by the cam. Rittinger considers it preferable to have an inelastic bumper and to suspend the table so that it will fall away from the bumper by its own weight. With elastic bumpers the number of strokes per minute should be from 12 to 16, and with inelastic bumpers from 40 to 50 for coarser stuff. For slimes, with inelastic bumpers, from 60 to 80 strokes per minute are required.

The Rotating Table (Fig. 106).—This machine, which is admirably adapted for the treatment of fine slimes, is in some respects analagous to the buddle in its operation and construction. The main differences are that the table rotates slowly under the feeding spouts, and the dressed ore, instead of being allowed to lie on the table, is washed off by a current of clean water as soon as the separation of the grains is effected.

The table consists essentially of an upright wooden or hollow iron shaft, to which a number of wooden arms, sloping either to or from the centre, are radially attached supporting the floor. The ore is fed upon the table by means of a current of water holding the particles in suspension, either on the circumference or the centre, according to the slope of the table.

The attachment of the arms to the upright shaft is accomplished by means of a casting, provided with openings into which the arms are inserted, and held in position by means of screws.

The inclination of this table is towards the centre, and this arrangement is generally preferred because the feeding apparatus is more easily reached when it is on the circumference, and because a larger surface is presented to the heavier

particles of ore, each of which may deposit itself without interfering with other particles.

To the radial arms short bits of plank are fastened by means of wooden pegs, which are employed in order that the surface

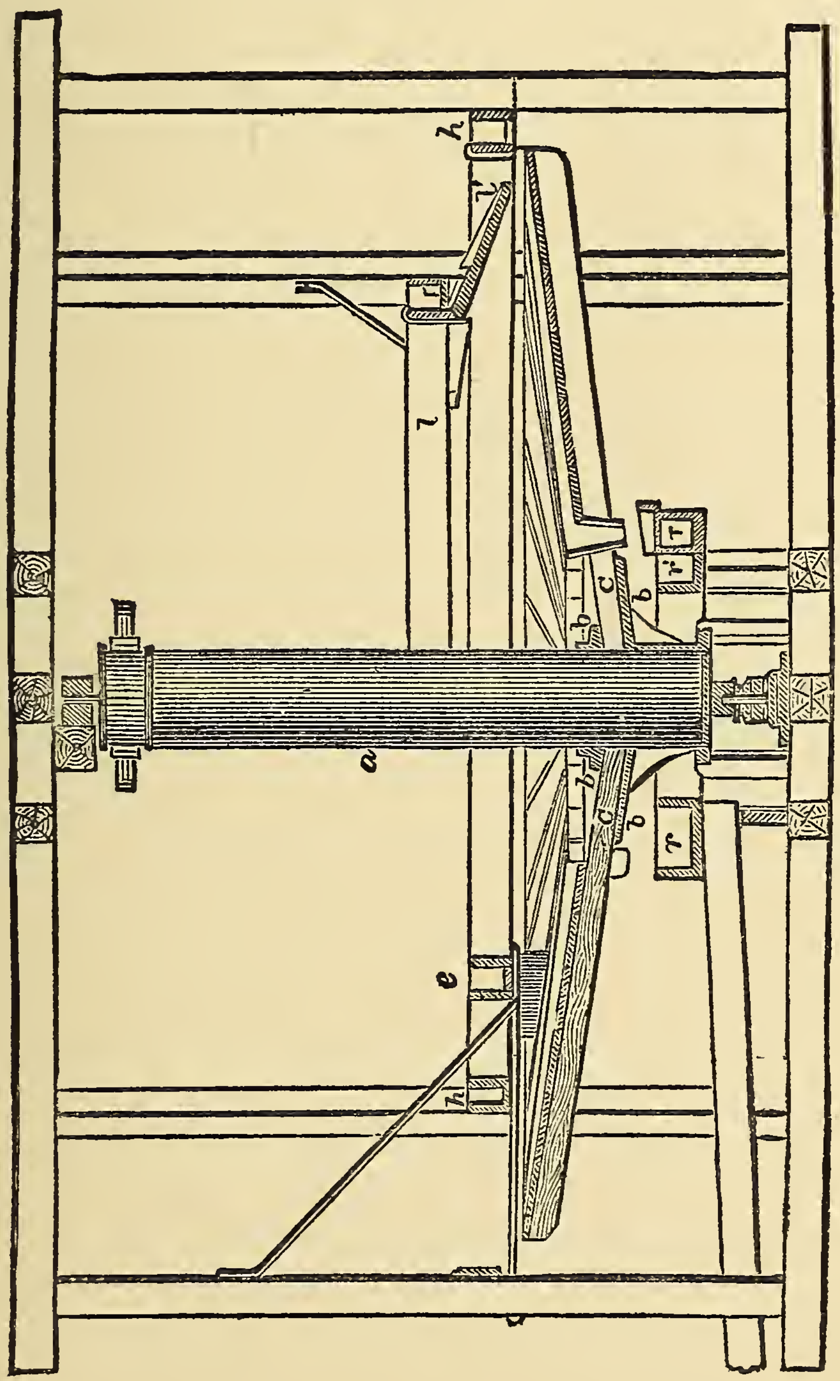


FIG. 106.—THE ROTATING TABLE.

of the table may afterwards be planed. The edges of these planks are smooth, and the joints are made tight by laying a piece of string or lampwick between them, and driving them tightly together before securing them by the pegs. The

exterior diameter of the table is 16 feet, and in the centre a circular space, 5 feet in diameter, is left open. Around the circumference of this circle is the discharge.

For treating fine slimes the inclination of the table should be 6 in. in $5\frac{1}{2}$ ft., or an angle of 5° $10'$ with the horizontal plane. After a smooth surface has been obtained for the table a number of small strips of wood (from 32 to 64) are nailed radially upon it, so as to divide the table into sections.

The watery mixture of ore is fed upon the table by means of feeding boards, which are so constructed as to cause the mixture to flow in a thin even stream. These boards are inclined to an angle of 20°, and their lower end is as broad as two of the sections on the table. The table is revolved very slowly, making only about six revolutions per hour, or one revolution in ten minutes.

There are four feeding boards, which are of course stationary, and from each of which flows a continuous stream. The motion of the table is so slow that by the time the sections, which were fed from the first feeding board come under the second the larger and lighter particles have been washed over the table and into the circular trough, *r*, situated beneath it, whence they are conveyed by suitable conduits. The same operation is repeated at each of the other feeding boards, and the heavy particles which were deposited upon the table from the first feeding board, together with similar particles from the other boards remain upon it till it is revolved past the last board.

Clear water is added in a stream sufficiently strong to wash the table clear of everything except the heaviest particles, which are finally washed off from each section when it reaches a certain point by means of a small flat stream of water under considerable pressure. The principal trouble with this apparatus appears to have been a choking of the compartments at their low and narrow ends. An extra circular trough, delivering water at these points, was found necessary but not altogether effective.

The Embrey Concentrator.—This machine consists essentially of an endless belt with flanges along its edges, supported on rollers in an inclined position, and having two movements, one a slow revolving motion up the incline, the other a vibratory shaking motion imparted by a crank or eccentric shaft to the frame carrying the supporting rollers. It differs from the well-known Frue vanner* in having the shaking motion parallel with the length and travel of the belt, while the Frue vanner has the motion at right angles thereto; in other words, the Vanner has a side shake, the Embrey an end shake.

The construction and action of the machine will be readily understood on reference to the illustrations (Figs. 101, 102, 103). G G is the main frame consisting of two sides, made up, as shown, of cap, sill, posts, and two braces, all bolted strongly together. These two side frames are joined together by cross sills and bolts at the bottom, and long bolts with collar shoulders inside at the top, as indicated in the drawing. The framework when erected makes a stiff support for the whole machine, and can be set on any ordinary floor, and blocked up at the front end if desired, for variations in the inclination of belt mentioned hereafter. Within the main frame a light shaking frame, F, is supported, on which the belt is carried. This frame, F, consists of two sides of wood with cross braces and bolts, and carries at each end a set of bearings in which the end rollers, A A^1, revolve. Between these end rollers a number of small galvanized-iron rollers, D D, are carried in bearings on the shaking frame, and with the end rollers form an inclined plane on which the upper or working surface of the belt, E, is supported. The belt, E, is made of rubber with canvas filling, has soft flexible rubber flanges around its edges to prevent the overflow of water from its surface, is 27½ ft. in length and 4 ft. in width.

The shaking frame (it will be seen) is supported on six legs or toggles, N N, standing in adjustable stirrups, *b b*, hanging on the main frame, and which are used sometimes for slight changes in inclination of frame or levelling of the belt across. The

* The Frue vanner is described in my "Metallurgy of Gold."

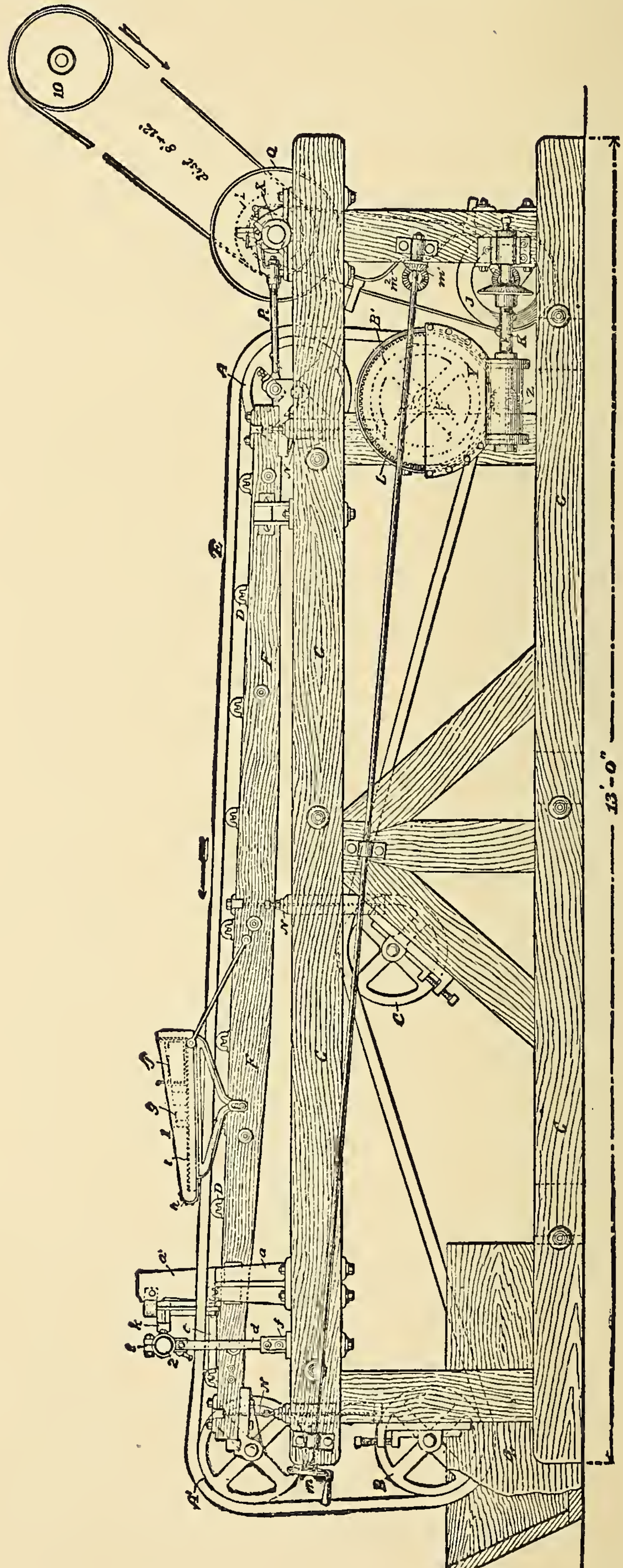


FIG. 107.—EMBREY'S CONCENTRATOR. Longitudinal Section.

shaking frame, F, is connected by short connecting rods, R R, attached to the lower roller bearings, to two eccentrics having about ¾-in. motion, and fixed on driving shaft, H. This driving shaft is fitted with tight and loose pulleys, I, which can be driven by a 3-in. belt from broad-faced pulley on any convenient shaft in the mill, or from a special countershaft put up above the machine. There are two small flywheels, Q Q, on the driving shaft, and also a cone pulley, J^1, which is connected by narrow leather belt to a corresponding cone pulley, J, on the shaft below. The speed of this lower shaft can be regulated as shown by the belt shifter, m^2, actuated through the small bevel gear and screw, m^1, by means of the hand wheel and rod, *m*, at the head of the machine. The motion of this lower shaft is communicated by bevel gear to the worm, Z, which in turn gives a slow revolution to the worm gear, L, on shaft of roller, B^1, around which the endless apron plies. The effect of the revolution of worm gear, L, with its attached roller, B^1, is to give a slow travel upward of the rubber apron, E, on which the concentration is effected, while at the same time the eccentrics on the driving shaft give the upper surface of the belt, through its supporting frame, a rapid but steady vibratory or shaking motion. The belt has therefore two motions, a slow forward motion and a rapid vibratory motion, the effects of which will be explained later.

The endless travelling belt is kept in position below by the three fixed rollers, B C B^1, of which C serves as a tightener, and in part as a regulator of the travel of the broad belt in keeping it straight on the shaking frame above. This latter object is also served at need by the adjustable bearings of the rollers, A^1 B, which, by tightening on one or the other side of the belt, causes it to travel to one or the other side of its course, and so admits of neutralising any tendency to run off the supporting rollers. The lower front roller, B, dips into a tank of water, 4, so that the belt in passing around it is submerged for a short time, so as to wash off any adhering concentrations which collect in the tank, 4.

On the upper side of the main frame, G, are four short cast-

iron standards, with projections inside, which serve as guides to

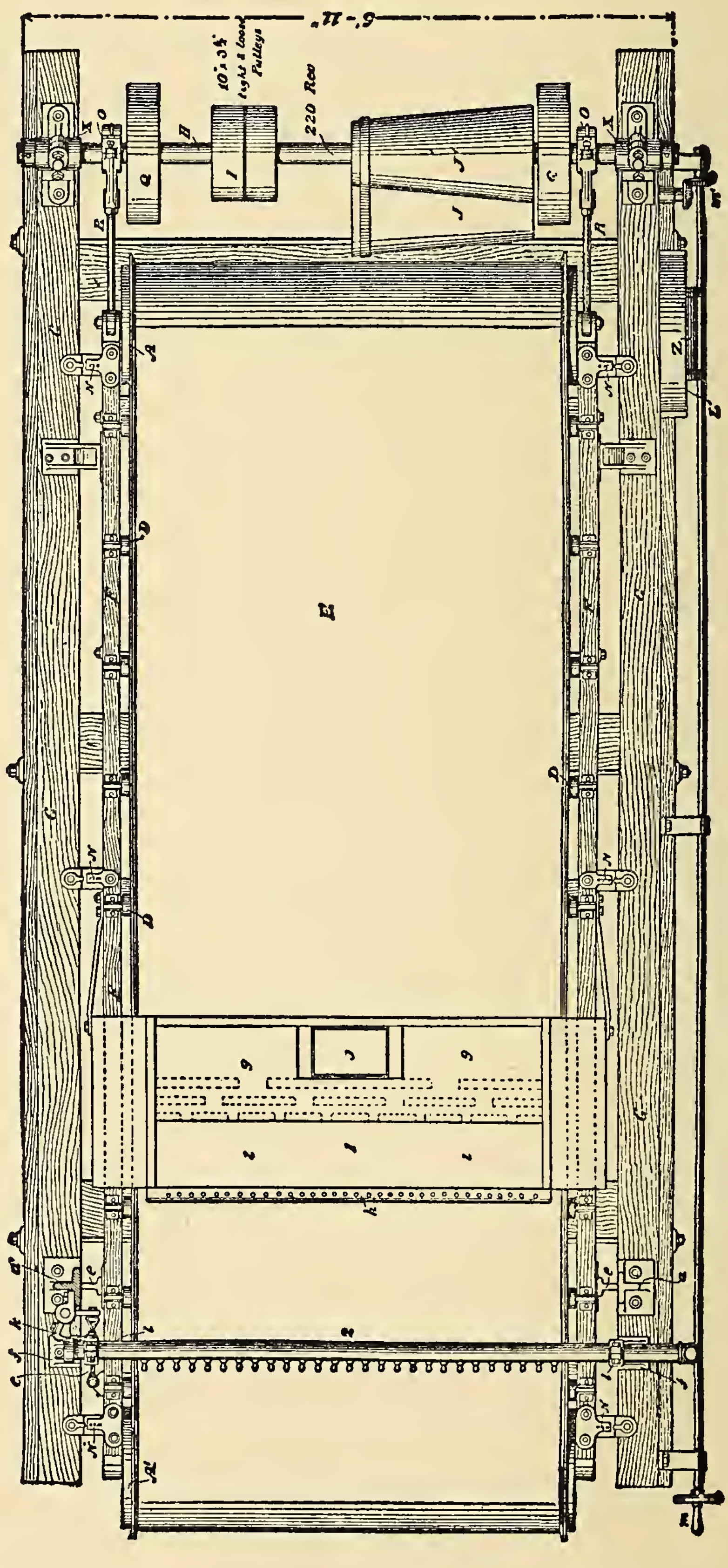


Fig. 108.—Embrey's Concentrator. Top View.

the sides of the shaking frame, so that its motion is simply in

the line of its length without any side play. One of these standards, a^1, is higher than the other three and serves a double purpose, its upper part being used as a support for a bell crank, *k*, which is attached by a strap connection, *c*, on the inside to

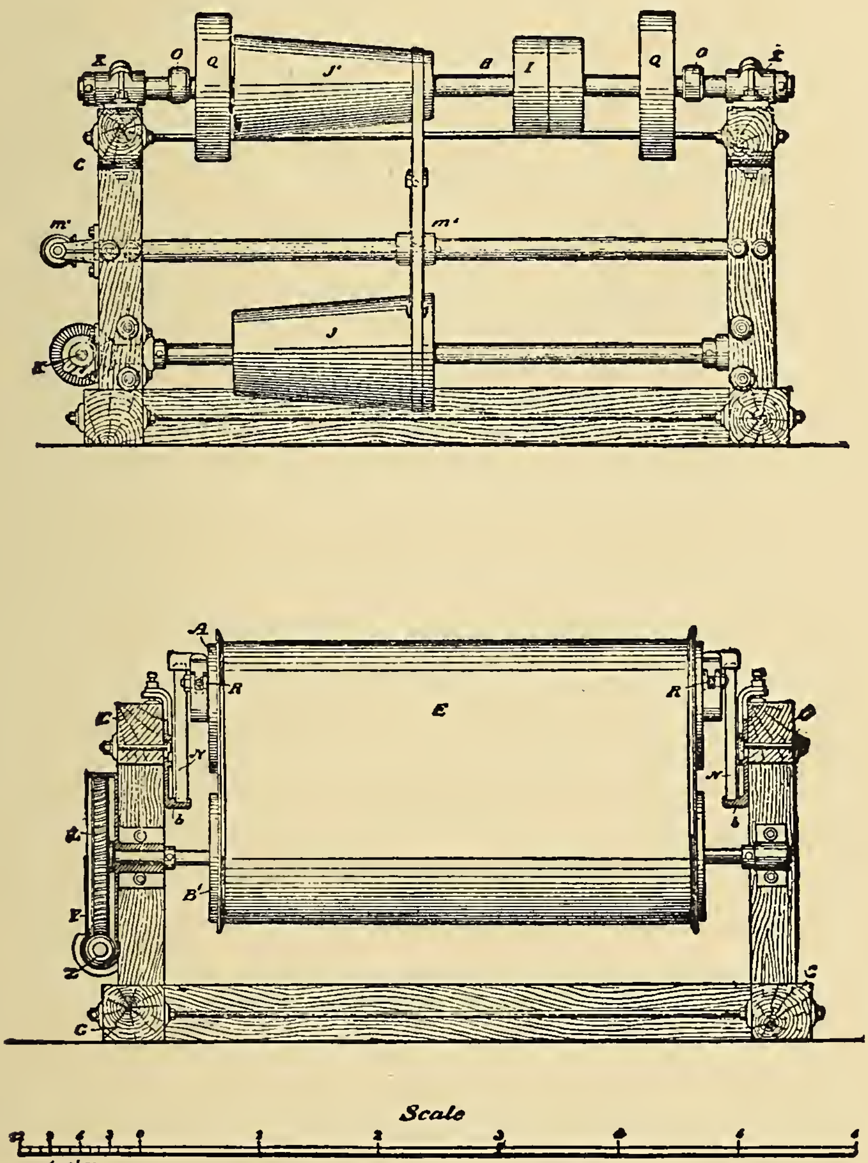


FIG. 109.—EMBREY'S CONCENTRATOR. End Elevation.

the shaking frame, F. The outer arm of this bell crank, *k*, is attached to one of two clamps, *l*, on the movable water pipe, 2, which stands on two spring legs, *d d*, bolted below in sockets, *f f*. This water pipe, 2, is fitted with a number of small jet cocks on its under side, and at one end is connected with

water supply by a flexible hose coupling to allow of its motion by the bell crank, *k*. The effect of the spring leg support and bell crank attachment to the shaking frame is to give the pipe, 2, a rapid shaking motion across the width of the belt, E, and coincident with the longitudinal shake given by the eccentrics, *o o*.

No. 1 is the ore distributor which receives the material to be concentrated, and spreads the same over the width of the belt. It is constructed as follows. The body of the distributor is a low open frame or box, with bottom, back, and two sides, but open in the front. At the front end there is a turned-up lip of sheet-iron, *h*, with a number of holes punched along its bottom. G is a board with strips of wood attached on its lower side, arranged as shown by dotted lines in the plan. These cleats or buttons are arranged so as to break up the current of water and pulp flowing around them, spreading the same evenly over the whole breadth of the distributor. At the back end of the spreader board, *g*, there is an opening into which is placed a small copper tank, *j*, which receives the flow from any launder or pipe of the pulp to be treated, and serves to retain any quicksilver or amalgam in the case of gold milling. In working gold ores, an electro silver-plated copper plate, *i*, is placed on the distributor, and under the spreader board, *g*. This plate with the copper tank, *j*, constitutes what is called an amalgam saver, and can be added to the machine when desired.

The arrangement of supporting stirrups and shaking frame, F, gives the latter a slight incline when the main frame is level on the ground. This inclination can be readily increased or diminished by blocking up one or the other end of the main frame, G. This blocking is best effected at the upper end, as not interfering then with the driving belt to the pulley at the lower end. Supposing the inclination of the belt to be adjusted, the operation of the concentrator is as follows.

The crushed ore—not coarser than 30-mesh and preferably 40-mesh size—in a small stream of water falls on to the distributor, 1, thence is spread evenly over the full width of the belt, E, and flows gently down the incline to the lower end of

the machine. The belt is subjected to a steady rapid shaking motion in the direction of its length, by the revolution of crank shaft, H, and at the same time it has a slow forward travel up the incline, communicated by the cone pulleys, J J′, and worm gear, L. The effect of this forward travel is to carry up all heavy particles of ore which settle on to the belt, in the passage of the pulp down its surface, assisted by the shaking motion. This shaking motion, by keeping all the pulp in gentle agitation, prevents the sand from packing on the belt in a mass —as it would do otherwise on so slight an incline with the small quantity of water used—and causes the heavier mineral particles to settle down to the belt, while the lighter waste rock is kept suspended in the flowing water, and passes off at the lower end as waste or tailings. The forward travel of the belt carries up to the water distributor, 2, all the heavy mineral particles, and here by the action of the many small jets of clean water falling on the belt, a final separation of the clean mineral from any adhering rock particles is effected. The mineral is carried past the jets of water by the revolving belt surface to which it adheres, and is deposited in the concentration tank, 4, where it is washed off by the passage of the belt through the water as shown in drawing. At intervals the concentrations are scraped out of the tank, 4. The lighter rock or waste and muddy water, constituting the "tailings," flow off the lower end of the belt into a suitable trough or launder, which carries them out of the building.

In working the machine, the concentration tank, 4, should have always sufficient water to properly immerse the belt as it passes around roller, B, and should be built with sloping front as shown, to facilitate the scraping out of the concentrates at intervals as they collect, to prevent accumulation and rubbing against the belt from a pile below it. The front of the tank is better with a flatter slope than shown in drawing. The main frame, G, should be firm and free from motion.

The inclination on the length of belt is usually about 3 in.; speed of crank shaft, H, 200 to 220. The lower speed is preferable if satisfactory work can be accomplished by regula-

tion of inclination and forward speed of belt. Having a fixed inclination and speed of shake, small jets of water are opened from distributor, 2, using as little water as possible, and then the chief means of adjusting the delivery of concentrates should be by the forward speed of belt through hand wheel, *m*. On some ores the shaking water distributor will be found a great improvement, on others the water pipe can be disconnected from the shaking frame and remain stationary. When the water distributor is in motion, less water is required to make clean concentrates than if stationary.

The material treated on the machine should not be coarser than 30 mesh, *i.e.*, should pass a wire screen of 30 holes to lineal inch. In case of coarser crushing, jigs should be used for all sizes above 30 mesh. The usual size for this class of concentrator is 40 mesh. In regard to quantity, the machine can be calculated on from 6 to 10 tons per twenty-four hours, and, therefore, either one or two machines are used to each 5 stamps, according to character of ore and closeness of concentration required.

The quantity of clear water used is from $\frac{3}{4}$ to $1\frac{1}{2}$ gallons per minute. The quantity of water coming on with the pulp is usually from $1\frac{1}{2}$ to 3 gallons per minute.

The Embrey concentrator, being a machine of complicated construction and nice adjustments, the most important points to be observed in running it may be mentioned here.

There are (1) perfect regularity in speed, in feed, and in water supply; (2) perfect cleanliness of machine, all wood work as well as iron work should be gone over every day, and all sand, dirt, or oil drippings removed; (3) proper attention to bearings, reasonable oiling, and taking up all the looseness or play; (4) avoidance of excess of water at head of machine as well as with pulp; (5) regulation chiefly by forward motion of belt, after once adjusting inclination and water supply. To attempt to regulate by water at head would require too much attention.

Dry Concentration.—Concentration of ores by means of

air is a problem which has received much study, and has been the subject of many experiments both in America and in Europe. The problem of using *compressed air* as an agent for the concentration of ores has been solved by Mr. Krom, of New York, who ascertained that air applied in intermittent impulses, similar to water in the wet jig, exhibits phenomena favourable to its employment as a concentrating medium.

The earlier experiments with air were confined to a continuous stream of air produced by a ventilator in a high, horizontal, and narrow conductor (two or three metres square), through which the dried material grains introduced from above were separated in a manner entirely similar to that of the horizontal stream of water, working in the settling pit according to the principle of free fall. But Krom starts with the fundamental idea of compressing the air by means of a kind of piston with quick successive strokes, and letting it work, thus compressed, with great rapidity through a low stratum of mineral grains, previously assorted according to size, which rests on fine sieves enclosed laterally in a narrow space.

Krom's Pneumatic Jig.—This machine, which has been devised for application of the principle of working just explained, is similar to the continuous hydraulic jig in outward appearance, and must be designated as an intermittent working air stream apparatus, in which, therefore, the dynamic effect of the living force produced by an upward jerk through the dense medium has a more energetic effect than by free fall. In the employment of air in puffs as a medium, its inferior density is a decided advantage. With water, the *vis inertia* of the mass of the medium to be moved prevents driving a jigger at more than 60 to 120 strokes per minute. Troublesome setting back of the water takes place, which is detrimental to the separation on the sieve, and to avoid which either the stroke and velocity must be diminished or a complicated arrangement be resorted to. The air, on the contrary, escapes at each stroke without setting back, and it is therefore possible

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to drive the pneumatic jigger 420 to 500 strokes per minute—very much more rapidly than the hydraulic one.

While the grains to be separated have to pass over a distance of one or two metres on the continuous hydraulic jigger, they have to pass over only thirteen centimetres of sieve length on Krom's machine. This apparatus can therefore, for the sake of concentrated efficiency, be more extended laterally without assuming troublesome proportions. As the adhesive effect of air compared with water is extremely small, it is evident that sands can be treated pneumatically which are very much finer than the finest treated on the hydraulic jigger. The dry sand forms a loose mass easily penetrated by the compressed air, while on the hydraulic jigger it is raised altogether in a closely adhering mass by the water, and even in falling hinders its own separation through mutual adhesion. While grains from 0·5 to 1 millimetre can scarcely be prepared on the hydraulic jigger, it is said that grains of 0·01 millimetre can be separated pneumatically.

The system, where tried in America, has given satisfactory results; and Mr. Stetefeldt, an eminent American authority on the metallurgy of precious metals, sums up his views on the Krom concentrator in the following words:—

"In examining the advantages of the Krom system of dry concentration compared with the wet method, we find—

"1st. That Krom's air jig effects a more complete separation of minerals of different specific gravities than the water jig.

"2nd. That material of greater fineness can be treated in the air jig than in the water jig.

"3rd. That in wet concentration the great losses occur in those machines which treat material too fine for the jig.

"4th. That the dust which results in the Krom system is of higher value than the original ore, and is a concentrated product. But besides this other conditions are to be considered. If the products of concentration are to be roasted and a portion of them have previously to be finer pulverized, it becomes necessary to get them perfectly dry. Now it is evident that it must be very much cheaper, and require a much simpler and

less bulky plant to dry the ore after it leaves the crusher than to remove the moisture from concentrations and slimes completely saturated with water. It furnishes dry dust ready for the roasting furnace, concentrations ready for finer pulverization, a concentrated product ready for smelting, purer and of higher percentage in lead than the water jigs. It also makes possible a better arrangement of the location of the dry kiln in connection with the mill."

The appliance is illustrated in the accompanying drawings, Fig. 110 being a perspective view and Fig. 111 a transverse sectional view. The machine is composed essentially of the following parts :—

A receiver, H, to hold the crushed ore; an ore bed, O, on which the ore is submitted to the actions of the air; the two gates, G G, one to regulate the flow of ore from the receiver, H, the other to determine the depth of ore on the ore bed; passage, C, in which the concentrated ore descends, and roller, R, to effect and regulate the discharge of the same; a fan, B, to give the puffs of air, a trip-wheel, T, lever, L, and spring, S, to operate the fan, and a ratchet-wheel, W, and pawl, P, to impart revolution to the roller, R.

The mode of operating the machine is as follows :—Ore is placed in the receiver, H, and the driving pulley set in motion. The cam-shaped trip-wheel, T, fixed on the opposite end of the pulley shafts works against the lever, L. By the alternate action of this wheel, forcing the lever in one direction and the spring which suddenly carries it back, the fan, B, is made to swing on the shaft, I, sending at each upward movement a quick and sharp puff of air through the ore bed, and lifting slightly the ore lying on it.

There are six projections upon the trip-wheel, so that the moderate speed of 80 to 90 revolutions per minute will give 480 to 540 upward movements of the fan in the same time, and a corresponding number of puffs of air to agitate the ore. This rate is sufficient to secure a steady motion of the heavy pulley, and yet not so fast as to produce any perceptible vibration, the machine working smoothly and easily. The ore bed is com

posed of wire gauze tubes, placed at distances from each other of $\frac{3}{16}$, $\frac{1}{4}$, $\frac{3}{8}$, and $\frac{1}{2}$ of an inch, according to the grade of ore to be concentrated, the finer requiring the tubes set nearer together, while the coarser allow the tubes to be placed further apart. The ore bed (situated in front of the fan, as plainly shown in the sectional view) is composed of these tubes. Their ends next to the fan being open, the air from the bellows enters and escapes through the top and sides of the tubes, agitating the ore that lies on them, and also that between them near the

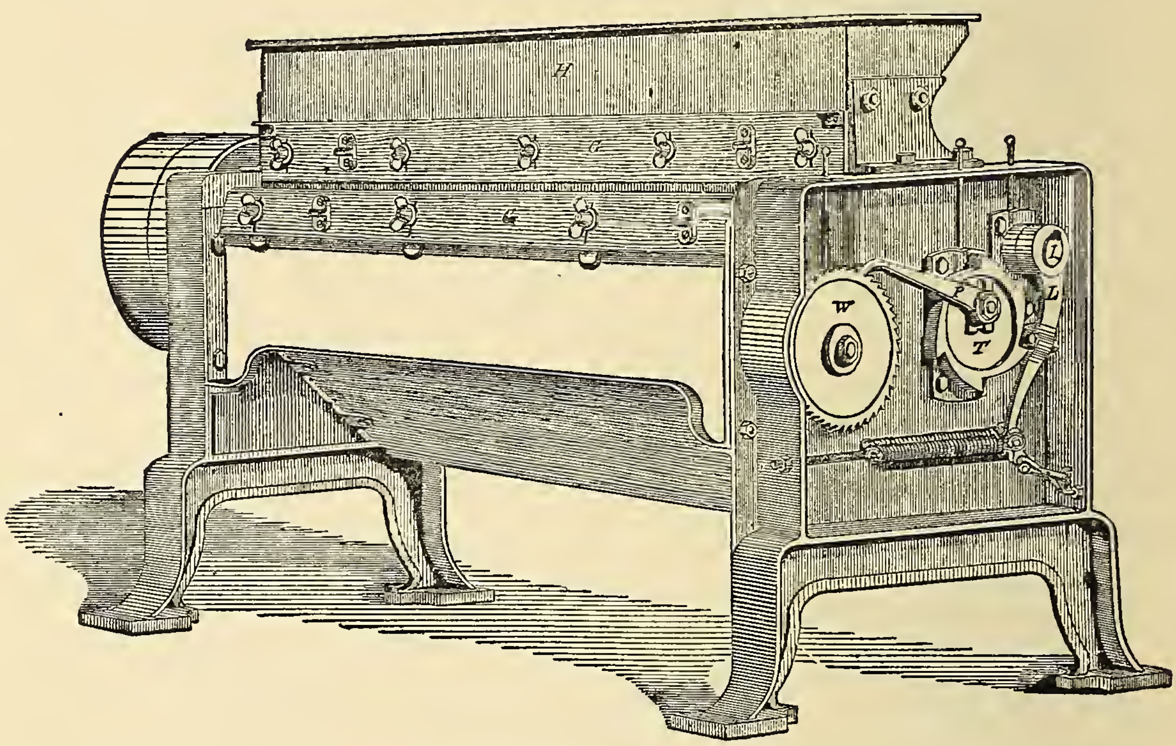


FIG. 110.—KROM'S PNEUMATIC JIG.

surface. The ore between the tubes rests on that immediately below in the passage, C, and sinks as fast as the roller, R, effects discharge. The tubes being open also on the bottom, any fine ore passing through the meshes of the wire gauze descends with the main body, C, thus preventing any liability of the tubes to fill up with fine ore.

The roller, R, is operated (as above mentioned) by means of the ratchet wheel, W, and pawl, P, and the latter being carried by a crank on the trip wheel, it follows that its speed is governed by the speed of this wheel, which also gives motion

to the fan, B. By this connection the fan, which effects the concentration, and the roller, which discharges the concentrated ore, are made to act in concert with each other. The importance of this feature will be apparent when it is remem-

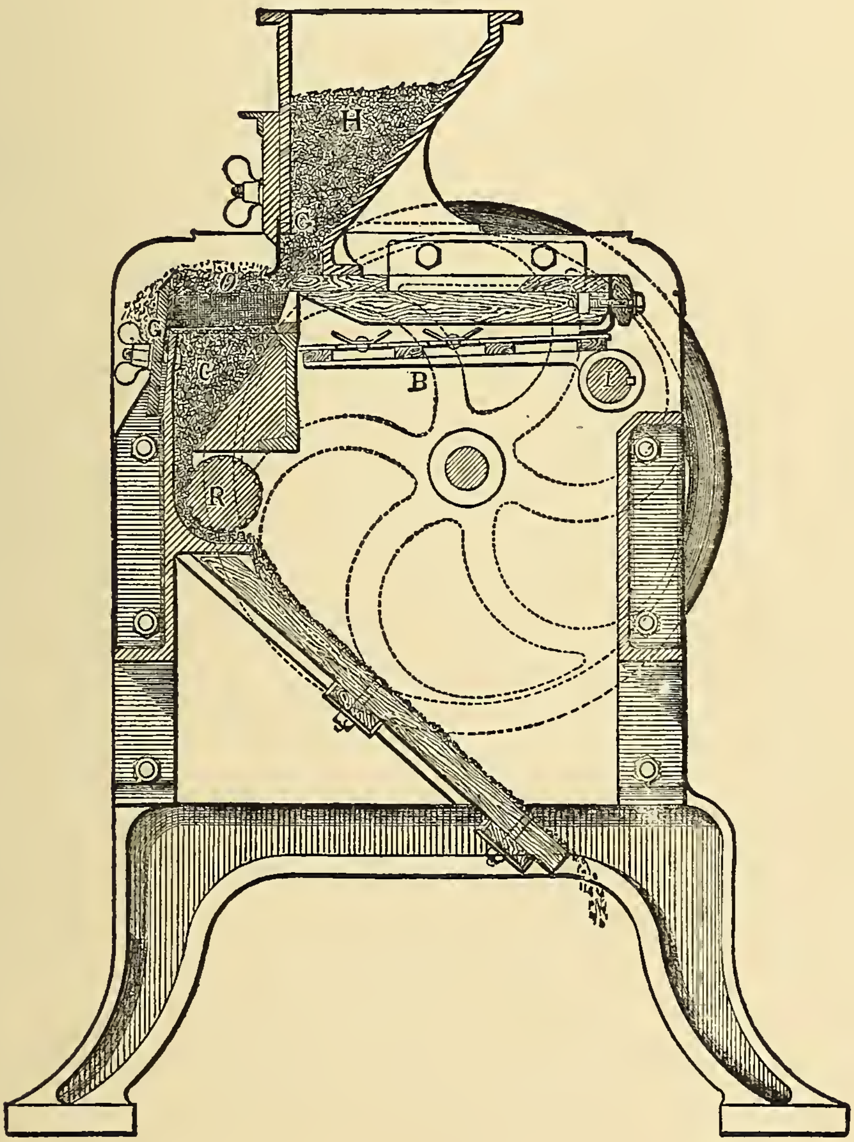


FIG. 111.—KROM'S PNEUMATIC JIG. Transverse Sectional View.

bered that the amount of ore concentrated in a given time depends on the rapidity of the puffs of air, so that the motion of the discharge roller, R, should be regulated to correspond with the speed of the fan.

The crank which carries the pawl can also be varied in length

so that the speed of the rollers may be regulated according to the richness of the ore. As already stated, the upper gate, G, governs the flow of ore from the receiver, H, to the ore bed, while the lower gate, G, regulates the thickness of the stratum of ore lying on the latter, as it is necessary to increase or diminish the depth of the bed of ore operated upon according to its coarseness or fineness. The finer the crushed ore, the thinner the stratum must be. The strap with its screw fastenings serves to prevent the roller attachment of the lever, L, from striking the body of the trip-wheel as it falls from each of the cam-shaped projections, and to regulate the extent of movement of the fan. That is, the strap must in all cases be so adjusted that the small roller working against the trip-wheel shall not strike at the foot of the cam, the cam serving in this manner to cushion the blow. Further, by tightening up or slacking off, by means of the screw fastening, the fan is carried in its vibration through a greater or less space, producing a stronger or lighter puff of air. The volume of the puff of air can thus be regulated to the requirements of different grades of ore.

Should the richness of the ore increase during working, and too large an amount collect on the bed, the air ceases to lift or agitate the material as much, and so a check is furnished to prevent loss in the tailings. No such check is possible in water concentration, because water moves practically as a solid and carries all before it.

CHAPTER XIII.

EXTRACTION OF SILVER BY LEACHING PROCESSES.

Treatment of Silver Ores by Solutions—Augustin's Process—Ziervogel's Process—Von Patera Process as improved by Russell—Mr. Stetefeldt's Account of Russell's Process—Lixiviation and Amalgamation compared.

There are several methods in use for the extraction of silver from the ore by solutions, depending in each case on certain chemical reactions. Of these methods, the oldest is—

Augustin's Process.—This is a method of extracting silver with salt water. When silver ores are roasted with salt they form chloride of silver, which is soluble in a concentrated solution of common salt, forming a double salt of chloride of silver and chloride of sodium. From this solution the silver is precipitated by copper, which again may be precipitated by iron. The remaining solution, after being purified from iron, sulphate of soda, &c., may be employed for dissolving a fresh quantity of chloride of silver.

The ore has to undergo an oxidizing roasting first, before the salt is added, resulting mainly in the production of chloride of silver. The oxidizing roasting volatilizes a great part of those substances which would otherwise readily combine with chlorine, causing a great consumption of salt, and if also converted into chlorides would be detrimental to the Augustin process; but if they are first oxidized they will not turn into chlorides on the addition of the salt.

Sulphuretted argentiferous compounds are best adapted for Augustin's process; if they contain metals combined with sul-

phur, arsenic, or antimony, which form easily caking combinations by the oxidizing roasting, considerable loss will ensue.

Many volatile chlorides are formed during roasting with salt, which by their volatilization carry away chloride of silver if roasted in ordinary reverberatories. Copper is turned into chloride of copper, which then partly volatilizes, whilst the remaining part is reduced to subchloride of copper, which dissolves in salt water together with the chloride of silver. When the chloride of silver is precipitated the subchloride of copper becomes again oxidized, forming oxychloride of copper, which being insoluble contaminates the precipitated silver. This is also the case with chloride of lead, which is soluble in hot salt water, and separates from the solution as it cools.

A previous roasting of antimonial and arsenical products, either with or without the application of steam—an extraction of the chloride of lead by means of hot water previous to the complete chlorination and lixiviation of the silver—a double roasting and a double lixiviation of speiss—are (all of them) suitable modifications of the process which have been partly employed in Hungary, but I know of no mine in America where this system is in use.

Ziervogel's Process.—This is a method of extracting silver with warm water, and is based upon the transformation of the silver contained in the substance under treatment into sulphate of silver, which is then lixiviated by hot water containing some sulphuric acid, and precipitated from this volution by means of copper. The formation of sulphate of silver is effected in the roasting process by the evolution of sulphuric acid in a gaseous form from sulphates which have been generated from other sulphides associated with the sulphide of silver.

Simple as this process may appear, it is very difficult to carry out practically, as it requires a very skilful manipulator to seize the exact moment when all the sulphates of the other metals are decomposed and none of the silver is.

The process is used in the treatment of copper matt. During the roasting care must be taken that all of the sulphates should

be decomposed (except the sulphate of silver), otherwise the silver will be precipitated by them. If all the base metal sulphates are decomposed there is also a danger that the silver sulphate will be decomposed, and it will be lost in the residues, as the silver oxides are not soluble.

If sulphide of iron predominates in the ore, and only a little sulphide of copper is contained in it, the formation of sulphate of silver will be imperfect, and rich residues will result in the subsequent lixiviation. The presence of a certain amount of iron in the matt is essential, otherwise too much time is required for the formation of sulphate of copper.

The existence of other foreign substances in the matt is injurious to the yield of silver—sulphide of lead and antimony for instance—as they cause a caking of the roasting mass. Zinc, antimony, and arsenic facilitate a volatilization of silver and also give rise to the formation of antimoniate and arseniate of silver, which is decomposed with great difficulty by sulphuric acid gas. If the decomposition takes place at a higher temperature, metallic silver will be formed, liable to volatilization.

The presence of metallic copper in the matt is also disadvantageous, as the silver in it cannot be extracted. This process is not fit for treating most of the silver ores, as they either contain antimony, arsenic, lead, and zinc in large quantities, or do not contain sulphide of copper. But the same is applicable to substances as pure as certain copper matt.

The extraction of silver by lixiviation, from ores which have been subjected to a chloridizing roasting (of which method the Von Patera process is an example) is based upon the fact that silver chloride is easily soluble in solutions of sodium or calcium hyposulphite, and that silver is precipitated from such solutions by an alkaline sulphide with regeneration of the hyposulphite salts.

The Russell Process.—The Von Patera process has been modified and improved by Mr. E. H. Russell; and in the following pages (for the substance of which I am indebted to

Mr. C. A. Stetefeldt)* a description is given of the process as thus modified.

If the ore, when treated in solutions of sodium or calcium hyposulphite, contains lead, a large portion of the latter is also dissolved, lead sulphate being soluble in hyposulphite solution. If at the same time copper is present, the sulphides precipitated from the solution contain silver, copper, and lead, a combination of metals not desirable for subsequent treatment.

Mr. Russell has discovered that lead can be completely separated from a sodium hyposulphite solution, as lead carbonate, by sodium carbonate or purified soda ash, without precipitating any copper or silver. After decanting the solution from the lead carbonate, silver and copper are obtained from it in the usual way. This method of separating lead prohibits the use of calcium polysulphide as a precipitant for the sulphides, because any calcium entering the regenerated lixiviation solution would also be precipitated as a carbonate with the lead by soda ash. Hence a sodium sulphide must be employed. A full investigation has demonstrated that this is by no means detrimental. Sodium sulphide and hyposulphite are more advantageously used in the lixiviation process than the corresponding calcium salts.

Another defect in the lixiviation process consisted in the necessity of a very perfect chlorination of the silver in the ore, because silver in any other combination, or in the metallic state, would be only imperfectly extracted by sodium or calcium hyposulphite.

Mr. Russell discovered that a solution of a double salt of cupreous hyposulphite and sodium hyposulphite, formed by mixing sodium hyposulphite with copper sulphate, exerted a most energetic dissolving and decomposing action upon metallic silver, silver sulphide, and its combinations with antimony and arsenic. Hence, if a charge of ore is first lixiviated with ordinary sodium hyposulphite solution to dissolve the silver chloride, and subsequently with cupreous hyposulphite—this solvent is called the extra solution—an additional amount of silver is extracted

* A paper read before the American Institute of Mining Engineers.

which would have been lost in the tailings by working according to the old method alone.

This process can also be introduced with profit to extract silver from ores without roasting, or after they have been subjected to an oxidizing roasting.

The sulphides of silver and copper, obtained as precipitates in the lixiviation process, are dissolved by nitrated sulphuric acid, the escaping nitric oxide being reconverted to nitric and nitrous acid according to well-known chemical principles. From the solution cement-silver is precipitated by metallic copper, and copper sulphate results by crystallization. A part of the latter is again needed for preparing the extra solution.

In the lixiviation of silver ores by means of a hyposulphite solution, which process was first proposed by Professor John Percy, two difficulties have heretofore been met with which have rendered the process inapplicable in many cases. These are first, the difficulty of producing bullion free from lead; and secondly, the necessity of a very perfect chloridizing roasting.

Mr. Russell has discovered the reaction that $PbCO_3$ is insoluble in a sodium hyposulphite solution, while the carbonate of silver and copper are soluble, which latter fact was known. Hence, if carbonate of soda is added to a hyposulphite solution containing lead, silver, and copper, PbCO, alone is precipitated. If a solution of pure $PbSO_4$ in sodium hyposulphite is so treated, the precipitation of the lead is so complete that H_2S gives no reaction in the filtrate. Upon these reactions is based the separation of the lead from silver and copper in Russell's lixiviation process. In effecting this on a large scale it is most economical to use the commercial soda ash. The latter, if manufactured by the old process, contains more or less Na_2S, and precipitates Ag_2S with the $PbCO_3$, whereby a product results rich in silver. In order to purify the soda ash, Mr. Russell makes use of the fact that $CuCO_3$ is soluble in a hyposulphite solution, and CuS is not. He dissolves the soda ash in water containing about $1\frac{1}{2}$ per cent. $Na_2S_2O_3 + 5$ aq., and then adds a solution of copper sulphate. Copper sulphide is precipitated, and the soda ash so purified

yields a lead carbonate retaining a trace of silver only. But the soda ash may also contain caustic soda. The latter if present in a hyposulphite solution has an injurious effect in extracting silver by lixiviation. In order to remove it, the soda ash solution is first boiled with sulphur, whereby sodium polysulphide and hyposulphite are formed, and then the sodium polysulphide is decomposed by copper sulphate as stated above.

The carbonates of iron, manganese, and zinc, and also of calcium, share with the lead the peculiarity of being insoluble in a hyposulphite solution. In well-roasted ores only traces of iron salts exist, and these, as well as the chlorides and sulphates of zinc and manganese, are removed by the wash water. Hence none of these carbonates are precipitated with the $PbCO_3$.

If lead is to be precipitated by this process the use of calcium hyposulphite is not admissible, neither can a calcium sulphide be used for the precipitation of the silver.

Mr. Russell calls the hyposulphite solution to which copper sulphate has been added the "extra solution," to distinguish it from the ordinary hyposulphite solution without copper. The discovery of the reactions of this extra solution constitutes, in its practical application, Mr. Russell's second and most important improvement in the lixiviation of silver ores.

At the Mount Cory Mill, Nevada, caustic lime is used as precipitant for lead from the lixiviation solution. In this case the lead is obtained as a basic hydroxide, together with gypsum and such impurities of the caustic lime as do not enter into the reaction. Cheapness and the possibility of using calcium pentasulphide for the precipitation of silver and copper are claimed as the advantages of this method.

The plant which Mr. Russell erected at the Ontario Mill for experimental purposes is capable of lixiviating a charge of 3 tons of ore in one tank. In his experiments the following facts have been ascertained.

It is just as effective and more judicious to leach the roasted ore with cold, and not with hot water, prior to turning on the hyposulphite solution. Hot water will dissolve a large per-

centage of AgCl, this being more soluble in hot than in cold brine. The first wash water which contains copper and some silver is conducted to tanks filled with scrap iron, where the copper, and with it most of the silver, is precipitated. Mr. Russell has determined in ten charges (of 2 tons each) the total amount of silver dissolved by the wash-water, and the amount recovered with cement copper.

Total amount of silver dissolved per ton . .	0·39 oz.
Amount of silver recovered with copper . .	0·28 ,,
Amount of silver lost	0·11 ,,

These figures will, no doubt, vary in other localities, where more or less salt remains undecomposed in the roasted ore, and acts as the principal solvent for the chloride of silver.

The Extra Solution.—After leaching with water the AgCl in the charge is first extracted with an ordinary hyposulphite solution, containing from $1\frac{1}{4}$ to $1\frac{1}{2}$ per cent. of the commercial $Na_2S_2O_3 + 5$ aq. This solution is made slightly acid with sulphuric acid, provided it contains caustic soda. After most of the silver chloride is dissolved the extra solution is turned on, to act upon the silver not present as chloride. This method of lixiviating has the best effect. Mr. Russell found that the extra solution is a much poorer solvent for silver chloride than ordinary hyposulphite solution. The solubility of silver chloride rapidly decreases with an increase of copper sulphate added to the solution. That this should be the case follows from the reactions taking place in the formation of cupreous hyposulphite.

The extra solution is made up by adding to a measured quantity of ordinary solution so much copper sulphate dissolved in the smallest quantity of water, that a standard extra solution is formed. By allowing the extra solution to sink through the charge, and pumping it up again, all particles of ore are brought in contact with it.

The peculiarity of the extra solution to part with a precipitate of a cupreous hyposulphite double-salt after some time makes it desirable to obtain this solution in a more permanent

form. This can be done by dissolving 18 parts sodium hyposulphite and 10 parts copper sulphate, each in a small quantity of water, mixing the solutions, allowing the precipitate of $2Na_2S_2O_3, 3Cu_2S_2O_3 + 5$ aq., to separate completely, decanting the clear solution of sodium sulphate and tetrathionate, and redissolving the precipitate (after washing it) in a pure sodium hyposulphite solution of from 1 per cent. to $1\frac{1}{2}$ per cent. concentration. This extra solution of constant strength is used for several ore charges in succession, until the cupreous hyposulphite is exhausted before it is turned into the precipitating tanks. Here the silver is best precipitated by sodium sulphide, from which any sodium sulphate which this reagent may contain has been removed by calcium sulphide. The pure regenerated solution of sodium hyposulphite is then utilised again for dissolving a fresh charge of $2Na_2S_2O_3, 3Cu_2S_2O_3 + 5$ aq. By this method a much better effect from the same quantity of copper sulphate is obtained than by the one first described.

In working on a large scale a larger percentage of silver is extracted than is shown to be soluble by laboratory tests. This is undoubtedly due to the prolonged time of the reaction, and to the use of ordinary and extra solution in succession.

An extra series of tanks has to be provided for the precipitation of the lead. The $PbCO_3$ settles in less than one hour, leaving a perfectly clear solution to be decanted into the silver precipitating tanks. Whenever the precipitate has accumulated in sufficient quantity it is collected by means of a filter press. The value of the lead carbonate will in many localities pay for the soda ash used in precipitating it. Besides, we should consider that the lead would otherwise have to be precipitated as sulphide by sodium sulphide. As this reagent is more costly than soda ash, whatever is realised for the lead is clear profit. The great advantage, however, is the absence of lead in the sulphides of silver and copper precipitated from the lixiviation solution.

Objections to Calcium Sulphide.—In Russell's process silver and copper have to be precipitated from a hyposulphite solu-

tion by a sodium sulphide. Calcium sulphide cannot be used, because any lime entering the lixiviation solution would be precipitated with lead as carbonate in effecting the separation of the lead. Calcium sulphide having been generally introduced in lixiviation works as a precipitant for silver, it might be argued that the necessity of abandoning this practice is a point not in favour of Russell's process for separating lead. Hence it is a question of importance to which hitherto metallurgists have paid little attention, to examine carefully if there are any tenable reasons for this preference, or if the practice is merely based upon a prejudice. In fact, the issue is a double one:—First, is calcium sulphide preferable as a precipitant for silver? Secondly, is a calcium hyposulphite lixiviation solution superior to one of the sodium salt?

The continued use of calcium sulphide gradually converts the original sodium hyposulphite solution into one of calcium hyposulphite.

In considering the first question the following points are involved, namely, (1) preparation of the sulphide solutions and their composition; and (2) their action and value as precipitants.

In preparing calcium sulphide, caustic lime is boiled in water with an excess of sulphur, so that the polysulphides CaS_4 and CaS_5 are formed. This excess of sulphur is necessary, because the lower calcium sulphides are not easily soluble in water, and this solution must be used rather concentrated. From the slight solubility of calcium hydrate in water it follows that this process must require considerable time. Sodium hydrate, on the contrary, is very easily soluble in water, and the sodium monosulphide and all of the polysulphides being equally so, the process must be completed very rapidly. In preparing, for instance, both solutions with the same amount of sulphur, and under otherwise equal conditions, the sodium solution precipitated after six hours' boiling seven and one-half times as much silver as the calcium solution. Continuing the boiling for seventy-two hours the calcium had reached its maximum of precipitating energy, but precipitated only 0.76 as much silver as the sodium solu-

tion after six hours' boiling. This demonstrates the wastefulness of the calcium process in sulphur and time.

It is evident that in the lixiviation process a certain quantity of hyposulphite must be lost, namely, in the wash-water, which has to be precipitated by itself, and is too dilute to be mixed with the normal solution, and also by decomposition in contact with the atmosphere. This loss has to be made good, otherwise the solution would get weaker, and finally refuse to dissolve any silver.

If a sodium sulphide solution, to be used for precipitating the silver, has been prepared as stated above, it contains a considerable quantity of sodium hyposulphite. For each part of silver precipitated 0.574 parts $Na_2S_2O_3 + 5$ aq., and for each part of copper 1·968 parts $Na_2S_2O_3 + 5$ aq. are added to the lixiviation solution. Hence in working high-grade ores, and especially if much copper is present, the lixiviation solution actually becomes more concentrated with continued use. The same reasoning does not hold good if freshly prepared calcium sulphide is used as the precipitant, according to what is stated below.

That a solution of calcium sulphide oxidizes rapidly at ordinary temperature, with formation of hyposulphite, is established by practical experience. Mr. Ottokar Hofmann, who used calcium sulphide as precipitant at the Silver King Mill, Arizona, states that the original sodium hyposulphite solution was used over a year and a-half, and that it increased in strength and volume, making it necessary to run a part of it to waste. In this case a large amount of copper and lead was precipitated with the silver.

At La Dura, Sonara, Mexico, the lixiviation solution was allowed to flow into the river by the stupidity of a labourer. A new supply of sodium hyposulphite could not have been obtained in less than ninety days, and in this dilemma Mr. Hofmann proceeded to manufacture a new solution from the calcium hyposulphite contained in the sulphide. The lixiviation wash water, containing the base metals, was precipitated with calcium sulphide, and the operation repeated until a calcium

hyposulphite solution of sufficient strength has been obtained to resume operations. I am sure this process would not have been successful with freshly prepared calcium sulphide, and that sodium sulphide would have answered the purpose much better.

Sodium Sulphide.— Sodium sulphide can be economically prepared by the following methods, namely:—

(1) By reducing sodium sulphate, at a high temperature, with carbon.

$$Na_2SO_4 + 2C = Na_2S + 2CO_2.$$

The disadvantages of this method are that the solution contains no hyposulphite, and that Na_2S in contact with air changes to hyposulphite with formation of caustic soda. The latter finally absorbs carbonic acid, and changes to carbonate.

$$2Na_2S + H_2O + 4O = Na_2S_2O_3 + 2NaHO.$$

In order to utilise such a solution in the lixiviation process it would be desirable to boil it with sulphur, and then expose it to the air. The oxidation of a solution of Na_2S_2 produces $Na_2S_2O_3$ only, while the higher polysulphides decompose in the same manner, but with precipitation of free sulphur.

(2) By melting sodium carbonate with sulphur.

Carbonic acid is liberated and sodium polysulphides and hyposulphite are formed. If the temperature is raised too high the hyposulphite is changed to sulphate.

(3) By boiling a solution of caustic soda with sulphur.

The reactions which take place in this case are expressed by the general formula introduced previously.

The choice of method will depend on local circumstances, namely, cost of chemicals, freight, &c. As the most simple in execution the third method recommends itself, and where freights are high it will also be the most economical. Regarding such a solution, it becomes a question of importance to have it so constituted that it produces a maximum effect with the smallest quantity of reagents consumed in its preparation.

If one of the heavy metals is precipitated by an alkaline polysulphite, RS_2, one equivalent of the latter precipitates not more than one equivalent of the former, S being liberated as free

sulphur. It seems, however, that there are exceptions to this rule if the metal exists in the form of a hyposulphite salt, and the alkaline polysulphide has been prepared in a certain way and is of a peculiar molecular constitution.

If sodium polysulphide is obtained by boiling caustic soda with sulphur, the equation—

$$6NaHO + 2\,(x + 1)\,S = Na_2S_2O_3 + Na_2S + 3H^2O,$$

shows that for 100 parts of caustic soda used the sodium polysulphide solution cannot precipitate more than 180 parts of silver as Ag_2S, according to the theory stated first above. Hence, 100 parts of commercial caustic soda, containing, say, 87 per cent. NaHO, the remainder being sodium carbonate and sulphate, would have a maximum precipitating energy of 156·6 parts of silver only. In preparing sodium sulphide from caustic soda of such quality Mr. Russell found that its precipitating energy for silver out of a hyposulphite solution was in many cases far in excess of the theoretical limit depending upon the original concentration of the solution in caustic soda.

Preparing the Precipitant.—The best method for preparing sodium sulphide, according to Mr. Russell, is as follows. The caustic soda is dissolved in an iron tank in its own weight of water. The solution should not fill more than one-fourth of the tank. Considerable heat is evolved, and in case the solution has not by itself reached a temperature of about 80° C. (this can easily be effected by using warm water), it is brought to 80° or 90° C. by a fire underneath the tank. For 100 parts of caustic soda 66 parts of pulverized sulphur are now gradually added. The temperature of the solution soon rises to about 145° C, and it foams up to two or three times its volume. In about four minutes the sulphur has dissolved and the reaction is completed. Upon cooling the mass solidifies; hence it is necessary to ladle it out and either dissolve it at once or cast it into moulds and preserve the cakes. If the sodium sulphide is dissolved, not in water but in lixiviation solution, no dilution of the latter takes place in precipitating the silver.

From the consumption of sulphur it follows that the sodium

sulphide is principally Na_2S_2, a combination to be considered as the most desirable one. Solutions so prepared show remarkably high precipitating coefficients, both for caustic soda and sulphur, ranging from 184 to 230 parts of silver for 100 parts of caustic soda, and from 275 to 345 parts of silver for 100 parts of sulphur. In the preparation of calcium polysulphide the best coefficients were from 98 to 132 parts of silver precipitated by 100 parts of sulphur consumed, and that with freshly-made solutions. Considering how much the latter deteriorate after some time, it is safe to assume that the precipitation of silver by calcium polysulphide requires about three times as much sulphur as is needed with properly prepared sodium sulphide. This item, the convenience of making sodium sulphide and the loss in hyposulphite caused by boiling caustic lime with sulphur, more than compensate the extra expense for caustic soda. If a hyposulphite solution is left to itself for a long time it deteriorates in a marked degree. It takes up oxygen from the air, and sulphates are formed.

Reducing the Precipitated Metals into Bullion.—Samples of sulphides, obtained from Ontario ore by Russell's process, if treated by the ordinary method (namely roasting and melting), yielded in one case bullion 874 fine, and for 100 parts of bullion 62 parts matt, assaying 2,720 oz. silver per ton, and 24 per cent. copper. In another case bullion 876 fine was produced, and for 100 parts bullion 76 parts matt were obtained, assaying 3,308 oz. silver per ton and 27 per cent. copper. A more complete roasting would have diminished the percentage of matt but produced bullion of inferior fineness. It cannot be denied that the handling of the sulphides is a weak point in the lixiviation process. In roasting such rich products a mechanical loss, and a loss in silver by volatilization, cannot be avoided. The by-product of rich copper matt, even if reduced in value by melting with scrap iron, is also a very undesirable feature. If oxidizing roasting is carried too far the material becomes more difficult to fuse, and the loss in silver by volatilization increases. In case the sulphides contain lead the bullion has to be cupelled, and it much copper is present more

lead must be added to produce fine bullion. That also in this process a loss of silver occurs is well known. Besides, litharge with copper is not a desirable by-product. Finally, it should be taken into consideration that the hearth of the reverberatory furnace in which the sulphides are roasted gradually absorbs a large amount of silver. Mr. Clark, the superintendent of the Bertrand Mill, states that he recovered from the hearth of such a furnace over $6,000 in silver, but that this by no means accounted for all the silver lost. To avoid all these difficulties, and to produce at the same time fine silver, gain the copper as a valuable by-product, and part the gold if it be present, Mr. Stetefeldt proposes the following process.

The sulphides collected by means of a filter press—Dehne's or Johnson's construction—are dissolved, without previous drying, in sulphuric acid with addition of sodium nitrate. No heating is necessary, the reaction being quite violent. If any gold is present it remains undissolved, together with some silver chloride, the chlorine being derived from impure sulphuric acid and nitre. The lead having been previously separated by the method already described (page 282), no lead sulphate can be formed. The sulphur gathers mostly in globules, and can be used again in the preparation of sodium sulphide. After complete decomposition has taken place the solution is drawn into a tank, and the silver precipitated by metallic copper. The cement copper gained from the wash water can be utilised for this purpose, and its silver extracted at the same time. It is preferable first to melt this copper, refine it, and cast it into plates of proper form. From the solution copper sulphates is obtained. The acid mother solution is used again for dissolving sulphides. A portion of the copper sulphate is utilised in preparing the extra solution.

In effecting the oxidation of the sulphides by nitric acid, set free from nitre by sulphuric acid, nitric oxide escapes. When the latter comes in contact with air and moisture, nitrous and nitric acid are regenerated, which are again ready to part with oxygen. This reaction, upon which the manufacture of sulphuric acid is based, can be utilised also in this case. It is

only necessary to dissolve the sulphides in a closed vessel and conduct the escaping nitric oxide to a coke tower, into which air is admitted below and a spray of water or sulphuric acid from above. The nitrated solution thus obtained is passed through the tower again until it attains a proper concentration, and is then used for dissolving fresh charges of sulphides. Of course, a portion of the nitric acid will be lost, and has to be made good by fresh supplies of nitre.

In case the process of regenerating nitric acid is used, the best *modus operandi* will be as follows. The first charges are dissolved with sulphuric acid of 66° B, and nitre, and the escaping nitric oxide, with a surplus of air, is conducted to a coke tower in which concentrated sulphuric acid is nitrated. In working with concentrated acids iron vessels can be used throughout, the nitric acid having the tendency to make the iron more passive. The solution is drawn into a tank, No. 1, where it is diluted to 58° B., and heated to about 110° C. It is kept at this temperature until perfectly clear, and is then drawn into a tank, No. 2, surrounded by cold water, where it is cooled rapidly. Here the silver sulphate crystallizes, and also copper sulphate whenever the solution has reached a sufficient concentration in the latter salt. The residue in tank No. 1 will consist principally of gold (if such is present in the ore), some silver chloride, lead sulphate (if the lead has not been completely precipitated previously), and sulphur globules, and may also contain some undecomposed sulphides. It is allowed to accumulate, and is worked by itself. After the sulphates in tank No. 2 have been completely strained from the acid mother solution, they are taken to a lead tank, No. 3. Sufficient water is added to dissolve the copper sulphate, and the solution is boiled with metallic copper until the silver sulphate is reduced. The reduction of silver sulphate is better accomplished by scrap sheet-iron (Lautenthal, Germany), or by ferrous sulphate (San Francisco, Cal.). Stetefeldt proposes to use metallic copper in order to gain a maximum of copper sulphate. The copper solution is crystallized in tank No. 4, and the cement silver is washed until it is sweet. The mother solution from tank No. 4 is used

in place of water for boiling and dissolving the sulphates in tank No. 3. We now return to the acid solution from tank No. 2. This is mixed with sufficient sulphuric acid of 66° B. to make good the loss incurred in the formation of the sulphates, and, after passing through the coke tower to be nitrated, is used for dissolving a fresh charge of sulphides. In order to avoid a loss of nitric acid as much as possible, it is best to charge the dissolving vessel with more sulphides than the nitrated acid is able to oxidize. The excess of sulphuric acid will keep the silver sulphate dissolved, if the solution is sufficiently hot, and the latter can be clarified in part in the dissolving vessel before it is drawn into tank No. 1. Operating in the way indicated no further dilution of the solution tank No. 1 is necessary or desirable. Whenever the residue in the clarifying tank, No. 1, has accumulated in sufficient quantity, it is returned to the dissolving vessel and treated with an excess of nitrated sulphuric acid. It will then consist principally of gold, silver chloride, and, perhaps, of lead sulphate. Both of the latter can be extracted by a solution of sodium hyposulphite, and the gold can be refined by cupellation with lead.

It will be seen that this process, if successfully executed, is not any more complicated or expensive than that of refining silver bullion in bars. On the contrary, the solution of the sulphides is more easily and rapidly effected, and no sulphuric acid is wasted.

A boiling of the sulphides with caustic soda, prior to their treatment by the process described, might be advantageous. The largest portion of the sulphur would in this way be extracted as sodium sulphide. It would not be necessary to filter or wash the sulphides before this first treatment—more or less lixiviation—solution in the sodium sulphide not being detrimental. Filtering and washing would take place after the boiling with caustic soda. Where freight is high it may be more economical to use concentrated nitric acid in place of nitre.

Lixiviation compared with Amalgamation.—The prin-

cipal points in favour of lixiviation, as compared with amalgamation, may be stated (says Mr. Stetefeldt) as follows :—

(1). In amalgamation the fineness to which the ore has to be crushed is determined by the capacity of the settler to work off coarse sands without loss of quicksilver. It is not practicable to use a coarser screen than No. 30 if the crushing is done by stamps. This is almost equivalent to sifting through a No. 40 revolving screen, if the crushing is done by rolls. In lixiviation pulverizing as coarse as possible is desirable. The limit of coarseness is determined by the roasting process. It depends upon the character of the ore, and, principally, upon the manner in which the silver-bearing minerals are distributed in the gangue.

(2). The original cost of the lixiviation plant is much lower than that of pans and settlers. A further saving is effected by a reduction in the size of the engine and boilers.

(3). In amalgamation the pans and settlers consume not less than $1\frac{1}{3}$ horse-power per ton of ore. The power for pumping solutions, &c., in the lixiviation process is merely nominal.

(4). In large mills the quantity of quicksilver in rotation represents a capital of from £6,000 to £8,000, while the stock of chemicals required for lixiviation does not cost more than one-tenth of this amount.

(5). With Russell's improvements the percentage of silver extracted by lixiviation is much higher than by amalgamation.

(6). Lixiviation by Russell's process requires a less careful chloridizing roasting. In many cases the salt may be dispensed with.

(7). The value of the lost quicksilver, and cost in wear and tear of the pans and settlers, amounts to more than that of the chemicals consumed in the lixiviation process.

(8). The lixiviation process permits the extraction of copper and lead as valuable by-products.

(9). The sulphides from the lixiviation process can be more easily converted into fine bars, and the gold parted, than this can be done with the bullion obtained in amalgamation.

(10). Amalgamation is invariably injurious to the labourer's health.

(11). Where gold-bearing silver ores have been roasted with salt, lixiviation extracts, in most cases, more gold than amalgamation.

(12). The possibility of lixiviating many so-called "free-milling ores," without previous roasting, including tailings, resulting from amalgamation of roasted or raw silver ores.

(13). The possibility of lixiviating with profit some classes of silver ores after they have been subjected to an oxidizing roasting only.

CHAPTER XIV.

MELTING AND ASSAYING.

Section I. Assaying Alloys and Ores of Silver by the Dry Method.—Process of the Assay—Classes of Ores for Assay, according to Mitchell — Scorification — Cupellation — Melting Furnace — Muffle Furnace.
Section II. Melting and Refining of Silver Bullion.—Melting the Amalgam for Bullion—Melting Room and Bullion Weighing Room at Consolidated Virginia Mine—Assaying of Silver Bullion—Assaying of Doré Bullion.
Section III. Assaying Silver by the Wet Method.—Process of the Assay—Measurement of the Salt Solution—Temperature of the Normal Solution of Salt—Calculating the Standard of Alloy—Assay of Silver Alloys containing Mercury—The Agitator.

Methods of Assaying.—To determine the quantity of silver in an alloy two methods are employed—the "dry" and the "wet." For determining the quantity of silver in an ore or mineral, only the dry method, or "fire assay," is employed in practice.

By the dry method the silver is combined with lead through fusion, and an argentiferous lead button is obtained. The lead is then removed by the interesting process of oxidation called *cupellation*, it being performed on a porous vessel made of bone ash, and called a *cupel.*

This process is subject to manifold causes of error, among which the main one is the loss which takes place through volatilization of the silver, and through its oxidizability.

Silver is volatile to a certain extent, and this is noticed under the action of the blowpipe, when it gives off reddish vapours. When heated together with other oxidizable substances, like arsenic, antimony, zinc, lead, it will volatilize, in certain proportions, with these metals.

Therefore, when silver is heated with oxide of lead, it will

oxidize to a certain extent, and in the state of an oxide will be absorbed into the cupel. This loss of silver has to be accounted for in bullion assays, where great accuracy is required. This want of accuracy in the dry method has, in a certain measure, led to the use of the wet method of silver assaying introduced by Gay Lussac, which is now practised in all establishments for determining the fineness of silver bullion, and is based on the readiness with which silver, when dissolved in nitric acid, is precipitated by chloride of sodium or common salt dissolved in water. Each of these methods will be dealt with in the present chapter.

I. Assaying Alloys and Ores of Silver by the Dry Method, or Fire Assay.

Process of the Assay.—The object sought is to obtain the silver and other metals that may be present in the form of an alloy with lead, which is afterwards introduced into the muffle, and cupelled in the ordinary way. All the oxidizable metals are thus removed, the silver remaining behind in a pure state—that is, free from all metals except gold, platinum, and any other non-oxidizable metals that may be present. Should appreciable quantities of such metals be known to be present, the dry method of assay is not applicable, and the wet method should be resorted to.

The preliminary operations to which minerals containing silver are subjected, and the form of cupels and other apparatus employed, having been fully described (in the analogous processes followed in the assaying of gold ores) in my work on the "Metallurgy of Gold," it is unnecessary to repeat them here.

Ores, according to their nature, are either (1) fused with a reducing flux, such as charcoal or black flux in conjunction with litharge, in order to produce metallic lead in which to collect the silver; or (2) fused with an oxidizing agent, such as litharge or nitre; or (3) scorified.

Ores of silver, in which the metals that are present exist in the form of reducible oxides, are commonly treated according

to the first of these methods. The proportion of litharge employed for this purpose, says Phillips, must be varied according to circumstances, as the resulting button of alloy should not be too rich, since in that case a portion of the silver is lost in the slag; nor too poor, as the cupellation would then occupy a long time, and a loss through sublimation be entailed. In ordinary cases, if 400 grains of ore be the quantity operated on, a button of 200 grains will be a very convenient amount for cupellation, and this may be obtained by the addition of 300 grains of litharge and from 7 to 8 grains of finely powdered charcoal. The whole is to be well mixed with 200 grains of carbonate of soda on a sheet of highly glazed paper, and afterwards introduced into a Battersea clay crucible, No. 8 (Fig. 112), of which it should not fill more than two-thirds the capacity. It is now covered with a thin layer of borax and heated in an ordinary assay furnace, care being taken to withdraw it from the fire—with the tongs shown in Fig. 113—as soon as a liquid and perfectly homogeneous slag has been obtained, as the unreduced litharge would otherwise be liable to eat through the pot and spoil the experiment. When it has sufficiently cooled, either the crucible is broken and the

Fig. 112. Clay Crucible.

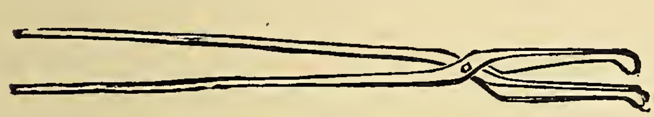


Fig. 113.—Crucible Tongs.

Fig. 114.—Ore Assay Mould.

button of alloy obtained is cupelled, or the fused mass is poured into a button-mould (Fig. 114).

When mineral substances other than oxides or carbonates are to be examined, the addition of charcoal or any similar reducing agent becomes, in many instances, unnecessary, as litharge readily attacks all the sulphides, arsenic sulphides, &c., &c., and oxidizes nearly the whole of their constituents, with the exception of silver, whilst a proportionate quantity of metallic lead is at the same time set free. The slags formed in this way contain the whole of the excess of litharge added,

and the button of alloy produced is subjected to cupellation in the usual manner.

All the directions given with regard to the assay of gold ores apply with equal force to the assay of silver ores. The treatment to which an ore is subjected depending to a great extent on its nature, it is well to make a preliminary assay, in order that in the actual assay only a certain amount of lead alloy may be submitted to cupellation. Mitchell recommends that all ores be subdivided into three classes, indicating the special treatment to which they should be subjected. These classes are—(1) ores which, on fusion with excess of litharge, give no metallic lead, or less than their own weight; (2) those which give their own weight, or nearly so, of metallic lead; and (3) those which give more than their own weight of metallic lead.

For the preliminary assay 20 grains of very finely pulverized ore are mixed with 500 grains of litharge, and placed in a clay crucible of such size that it is only half full. After being carefully warmed, it is rapidly heated in a bright fire, and when the mixture is thoroughly fused, the pot is removed, cooled, and broken. The ore belongs to the first, second, or third of the above classes, according as the resulting button weighs less than, the same as, or more than 20 grains, the weight of ore originally taken. Now 200 grains of lead alloy is a convenient quantity to cupel; hence, in the actual assay, bodies of the second class require simple fusion with litharge and a suitable flux; those of the first class require the addition of such a quantity of a reducing agent that the final button may weigh 200 grains; and those of the third class, such a proportion of the oxidizing agent as shall oxidize all lead in excess of 200 grains.

An excellent reducing agent is argol (commercial cream of tartar); and nitrate of potash is the oxidizing agent usually employed. The former is capable of reducing about six times its weight of litharge, and the latter of oxidizing about four times its weight. These data suffice for determining the quantities required for assaying ores of the first and third classes

respectively, when the results of the preliminary assays are known.

Assay of Ores.—*Class No. I.* (*Mitchell*).—Such ores as contain oxide of lead, carbonate, phosphates, are fused with a reducing flux. When ores contain a large proportion of sulphur, antimony, or arsenic, they are roasted. In making assays, special attention must be paid to the proper quantity of fluxes, so that the silver lead produced be not too rich, or that too great a proportion of lead be reduced. If the silver lead be too rich, much of the precious metal may be lost in the slag, and if too great a quantity of lead be produced, silver is again lost, owing to the long exposure to the fire during cupellation; and indeed this is the most fruitful cause of loss, for more is lost in this manner than by having too little lead produced. One part of charcoal reduces 30 parts of lead, and one part of black flux reduces about one part of lead.

FIG. 115.—MORTAR.

In making assays of this class of ores I generally proceeded as follows:—I prepared the fluxes, which were kept in closed stone jars. The litharge and carbonate of soda were carefully sifted. I used the following proportions, which were weighed out on a pulp scale after the ore had been pulverized in the mortar (Fig. 115):—

240	grains of	ore, or half an ounce.
300	„	carbonate of soda.
500	„	litharge.
25	„	argol or a proportionate quantity of charcoal.
100	„	fused and pulverized borax.
100	„	salt.

(The salt imparts a great fluidity to the fused slag, and rinses the pot well when pouring the mixture into the mould.) The ore, carbonate of soda, litharge, argol, and borax having been well mixed before their introduction into the pot, they were then covered with the salt. After fusion the pot was taken out with tongs (Fig. 113), cooled, and then broken. The

button so obtained had to be hammered into a cubical form, and should weigh 200 grains more or less; or the assay might be poured into a mould (Fig. 114). Assays are always made in duplicate.

Class No. II. (*Mitchell*).—In this class of ores—namely, the ores which give their own weight, or nearly so, of metallic lead—if the preliminary assay of the sample furnishes 20 grains, more or less, of lead, then the assay proper is made by mixing

	240	grains of	the ore.
	240	„	carbonate of soda.
	1,000	„	litharge.
and covering with	100	„	borax.
	100	„	salt.

After fusion the assay is poured into the mould, or the crucible is cooled and broken.

If ores contain sulphur the litharge effects a strong oxidizing power on the substance, and there ought to be sufficient litharge present to drive away all sulphurous matter, so that the slag may not contain the least trace of silver.

To decompose iron pyrites 50 parts litharge are required; arsenical compounds, zinc blende, sulphuret of antimony, copper pyrites, grey cobalt, require 25 to 40 times their weight; sulphuret of bismuth requires 10 parts; galena, sulphuret of silver, require 5 parts of litharge. These proportions are for the pure minerals; of course if they carry much gangue the litharge will decrease accordingly. As very large lead buttons would be obtained in this way, part of the oxidation can be performed by means of nitre. By employing suitable proportions of nitre and litharge all the silver contained in oxidizable minerals may be extracted, and any quantity of lead required may be thus alloyed with it.

Mitchell states that it requires 2½ parts of nitre to completely oxidize iron pyrites, 1½ for sulphuret of antimony, and 1 for galena. The sulphur in the ore combines with the nitre to form sulphate of potash, forming a slag.

Class No. III. (*Mitchell*).—For ores which give more than

their own weight of metallic lead, the assay mixture is as follows:—

240	grains of	ore.
240	,,	carbonate of soda.
1,000	,,	litharge.
50	,,	nitre.
100	,,	borax.
100	,,	salt.

To fuse this mixture I employ a No. 10 Battersea crucible,* as in this assay the crucible must be larger than in the two preceding cases. The mixture should not fill it more than one-third, as there is a considerable action set up between the oxygen of the nitre and the sulphur or arsenic, or any other substance that may have to be reduced in the ore; for in fact the nitre does not only oxidize the lead, which sulphur might have reduced, but oxidizes its equivalent quantity of sulphur, or whatever other reducing substance there may be in the ore, so as only to leave a sufficient amount to reduce 200 grains of lead. Care must be taken to employ the proper proportion of nitre —which sometimes has to be determined by preliminary assay —as an excess of nitre may oxidise all the lead and carry it into the slag, and no metallic button may be obtained at all; but fusion with nitre is to be recommended, as it produces a clean lead button. The metallic button obtained after fusion is submitted to cupellation.

Ores of this class may also be roasted at a low heat in the muffle on a roasting dish and then fused as above, and the quantity of nitre can then be reduced, according to the degree to which the roasting has been pushed.

Scorification.—This process has been described in the "Metallurgy of Gold," but I will add some remarks here as to its application to silver ores.

The button obtained from a scorification assay ought to be

* In my experience of crucibles I have found the Salamander brand of plumbago crucibles most reliable for the melting of bullion, and the crucibles of the Morgan Company, of Battersea (London), the best for ore assays; and, moreover, the Morgan Company are always ready to carry out any practicable suggestion for the improvement of their manufactures.

as ductile as ordinary lead ; if not, it cannot be cupelled, and must be submitted to a fresh operation.

It is better to push the scorification to its greatest extent, because less silver is lost than when a large button is cupelled. The button of lead remaining ought to weigh about 200 to 300 grains. The length of time a scorification takes is from half an hour to an hour. The scorifier is rendered less permeable to the litharge by being rubbed inside with chalk or red ochre. Scorifiers (Fig. 116) were formerly made in the laboratory during spare time by an assistant, but they are now to be had of manufacturers and of a reliable quality. The running sizes are 2¼ to 2½ inches in diameter, but the tendency is rather in favour of an increase to 2¾ and 3 inches. This mode of assay has an advantage over the crucible assay in its requiring no preliminary assay, but that advantage is counterbalanced by the fact that no more than 50 grains of ore can be operated on in one scorifier, and that good and trustworthy results cannot be obtained by this method unless four scorifiers are employed for each assay, so that in all 200 grains of ore may be em-

Fig. 116.—Scorifier.

Fig. 117.—Scorifying Tongs.

ployed. When very rich copper ores, however, have to be assayed for silver, the scorification process is very useful, as in the crucible operation much copper is reduced with the lead, so as to require a very large quantity of lead for its conveyance as oxide into the cupel.

The assay in the scorifier is executed by weighing out—

600	grains of	granulated lead.
50	,,	ore.
50	,,	borax.

The mixture is placed in the scorifier (Fig. 116) in the red-hot muffle with the scorifying tongs (Fig. 117), and when the surface of the metal is quite covered with fused oxide pour the contents of each scorifier into one of the hollows of the mould. When the mass of slag and metal is cold, separate the latter from the former by means of the hammer and anvil. Hammer

the metal into the form of a cube and reserve it for cupellation. In the scorification assay, the silver in the ore is taken up by the lead, and the superfluous lead and base oxides are slagged off.

Basic ores, containing much calc spar, baryta, fluor spar, require more flux, like borax, and an addition of silica. Too large an addition of fluxes must be avoided, as in covering the metal bath the oxidization of lead cannot take place, which prevents the decomposition of the sulphides, in consequence of which oxysulphurets containing silver remain in the slag.

If a larger addition of borax is needed, as in the presence of much tin, zinc, lime, &c., a part of it must be added afterwards, before the final strong heat is applied, especially also in the presence of a large quantity of iron.

Ores containing zinc blende, pyrites, copper, nickel, and cobalt compounds require a large percentage of lead, from twelve to thirty times their weight.

The scorifiers are introduced in the muffle when red-hot, and the lead will enter very soon into fusion, and will be covered by the lighter ore which undergoes roasting.

After twenty to thirty minutes, the surface of the assay has become smooth, and the fused slag surrounds the periphery of the molten bright lead in the centre, which is seen working off crystals of litharge as soon as the door of the muffle is opened. Full access of air is now given to the interior of the muffle, while the draught of the furnace is closed, and after about ten to fifteen minutes the oxide of lead and slag will cover the metal bath. The door of the muffle is again closed, the draught opened so as to give a final heating and render the slag fluid, and after five minutes the assay is ready to be poured.

Cupellation.—This process (which also has been described in my "Metallurgy of Gold") is based on the property of lead and other oxidizable metals present in the lead button being absorbed by the cupel, leaving a silver button behind on the cupel, and in case there is any gold in the ore the same is alloyed with the button. The scoriæ produced during cupellation are absorbed by the bone ash of which the cupels are made, and the lead partly volatilizes and is partly absorbed as litharge by

the cupel. Other oxidizable metals, which alone would not soak into the cupel, will become absorbed when fused together with lead or bismuth.

The cupels are heated to redness before the buttons are introduced. When the cupels are charged the muffle door is closed till all the buttons are in a perfect state of fusion; when this is the case the muffle door is slightly opened and air is allowed to pass into the furnace, and the metallic bath is then *uncovered*, as it is termed. On the admission of air the same becomes lustrous, and is covered with luminous and iridescent patches which move along the surface of the bath, gliding down the sides till they reach the edges of the bath, to become absorbed by the cupel. These patches are fused litharge, and impart to the bath the luminous appearance.

At the same time a vapour rises from the cupels which fills the muffle and is produced by the vapour of lead burning in the atmosphere. An annular spot is soon observed on the cupel around the metal, and this spot increases incessantly until it has reached the edges.

The phenomenon of *brightening*, or *fulguration*, and of *vegetation*, or *spitting*, has been fully explained in the assay of gold. The cause of this last effect seems to be that when the fused buttons are suddenly exposed to the cold air, the silver solidifies on the surface, whilst that in the interior remains liquid. The solid crust, contracted by cooling, strongly compresses the liquid interior, which opens passages for itself, through which it passes out, and around which it solidifies when in contact with the cool air. But it sometimes happens that, when the contraction is very strong, a small portion of the silver is thrown off in the shape of grains, which are lost.

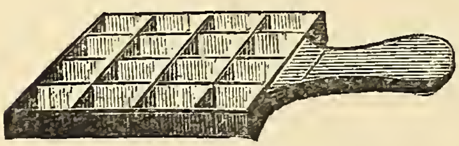


Fig. 118.—Cupel Tray.

After brightening, the cupels must be left for a few minutes in the muffle, and are gradually drawn to the front, so as to cool them gradually. When withdrawn they are placed on the cupel tray (Fig. 118).

The cupellation should not be conducted at either too high or too low a temperature. When the muffle is too hot the

fumes are hardly visible, and when the temperature is too low the fumes are thick and heavy, and the litharge can be seen not liquid enough to be absorbed, forming lumps and scales about the assay. When the degree of heat is suitable the cupel is red, and the fused metal very luminous and clear. At the end of the cupelling operation the heat should be increased, so as to remove the last traces of lead. If a large quantity of base metals are present, which are not being absorbed into the cupel, but accumulate round the liquid mass, the assayer had better add a fresh portion of lead and increase the heat at once before the assay "fouls."

In an assay which has succeeded well, the resulting silver button is well rounded, white, clear, crystalline below, and readily separated from the cupel.

As the litharge contains silver, the same is ascertained beforehand by cupelling a certain quantity of the reduced litharge, comparative reductions being made on the assay results. In cupellation there is always a loss of silver caused either through volatilization, or through oxidation, or through absorption of silver globules into the cupel. The last is the most important cause of loss, and increases with the coarseness of the cupel; and this has been demonstrated by extracting the litharge from the cupel and assaying the same for silver.

The cupels should be always proportionately large enough for the absorption of the button, and it is estimated that a cupel will absorb its own weight of lead.

The various metals found in an alloy, which can be submitted to cupellation, scorify in proportion to their oxidizability. Those most oxidizable scorify with the greatest rapidity, and *vice versâ,* so that those which have the greatest affinity for oxygen accumulate in the first portions of litharge formed, which, by that means becoming less fusible, sometimes lose the property of penetrating the cupel; hence the reason why cupellations always present more difficulties at the commencement of the operation than towards the end, when the litharge forms in nearly pure oxide of lead, which can contain only oxide of copper.

The amount of lead present should be greater in proportion

as the quantity of copper is more considerable ; but in cupelling an alloy of silver and copper, the same amount of lead cannot safely be added as in treating an alloy of copper and gold, as the greater volatility of the silver would cause a considerable loss of that metal before the whole of the lead could be absorbed.

The amount of lead necessary to effect the proper cupellation of various alloys of silver and copper is as follows :—

Standard of Silver.	Ratio of Weight of Lead to Weight of Assay.	Ratio of Weight of Lead to Weight of Copper Present.
1,000	1	—
950	3 to 1	60 to 1
900	7 „ 1	70 „ 1
800	10 „ 1	50 „ 1
700	12 „ 1	40 „ 1
600	14 „ 1	35 „ 1
500	16 or 17 „ 1	32 or 34 „ 1
400	16 „ 17 „ 1	27 „ 28 „ 1
300	16 „ 17 „ 1	23 „ 24 „ 1
200	16 „ 17 „ 1	20 „ 21 „ 1
100	16 „ 17 „ 1	18 „ 19 „ 1
Pure Copper	16 „ 17 „ 1	16 „ 17 „ 1

Makins, however, points out that there are great difficulties in working fine silver with less than three times its weight of lead, and he considers that results obtained on alloys very rich in silver are not satisfactory when a smaller quantity is used. He employs six times its weight for the assay of metal of the English standard (925 parts to 1,000), and increases the proportion for coarser varieties.

In the above table the numbers in the second column, which express the quantity of lead necessary to be added in each case, are multiples of the weight of alloy on which the operation is performed. Hence, in the third line, for example, in which the alloy consists of 900 of silver and 100 of copper for each grain of alloy taken, 7 grains of lead will be necessary ; and as each grain of alloy contains only one-tenth of its weight in copper, it follows that the ratio of the lead to the copper is as 70 to 1.

It will be remarked also that here, as in the table given for an alloy of gold and copper, the proportion of lead to be employed for a silver alloy below the standard of 500 remains constantly the same.

When other metals besides lead and silver are present in an alloy, the cupel usually affords indications, from which it is generally possible to judge of their nature, and roughly of the amount in which they exist. The following are the results of experiments made in order to ascertain the influence of the several metals on the nature of the resulting button, and the action on the cupel; but the appearance due to each metal is so often masked by the presence of another, that the indications given cannot be considered of much practical value.

Aluminium.—Yellowish stain, with scoriæ which float on the surface of the button.

Antimony.—Light yellow slag, which sometimes cracks the cupel.

Arsenic.—Sage green stain.

Cadmium.—Black ring near the top of the cupel.

Chromium.—Dark brick-red stain.

Cobalt.—Dark green scoria and greenish stain.

Copper.—Dark brown or green colour.

Iron.—Dark red-brown stain at commencement of operation, leaving a dark ring on the cupel.

Lead.—Straw or orange-yellow colour.

Manganese.—Dark bluish-black stain and corrosion of the cupel.

Nickel.—Dark green scoria and greenish stains.

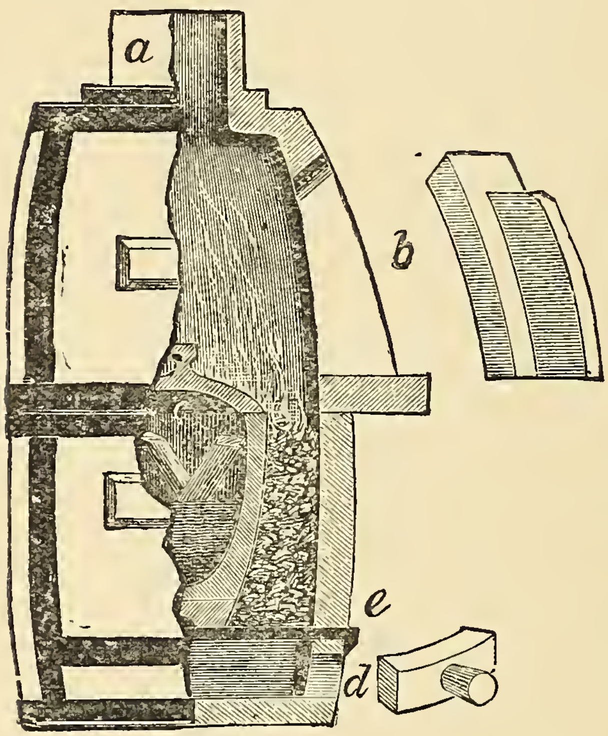
Palladium and Platinum.—Green stain and very crystalline button.

Tin.—Forms a grey slag.

Zinc.—Yellow ring on cupel; the metal burns with a brilliant flame, gives out copious vapours, and corrodes the cupel.

The loss by volatilization and absorption in the cupel being much greater than in the assay of gold, it is necessary to ascertain this loss by "check assays." These check assays are made on fine silver, side by side with the metal under examina-

tion, in the manner which will be found recommended in the chapter on the assay of gold in my work on the "Metallurgy of Gold." Since, however, the loss is much greater, and varies considerably throughout the muffle, it is necessary that a great number of check assays be made. In the Royal Mint nine checks are worked in a fire of 45 assays, or one in each row, and a correction is applied to each assay deduced from the checks in its immediate neighbourhood. Considerable judgment is required in applying such corrections to the weights obtained. The amount of the correction may be as much as 10 parts in 1,000 when assaying bullion, and the loss of silver increases with the weight of lead present.



a, socket for fixing on iron chimney.
b, door for putting in crucible and for coking.
c, crucible.
d, door for regulating the draught.
e, iron grate.

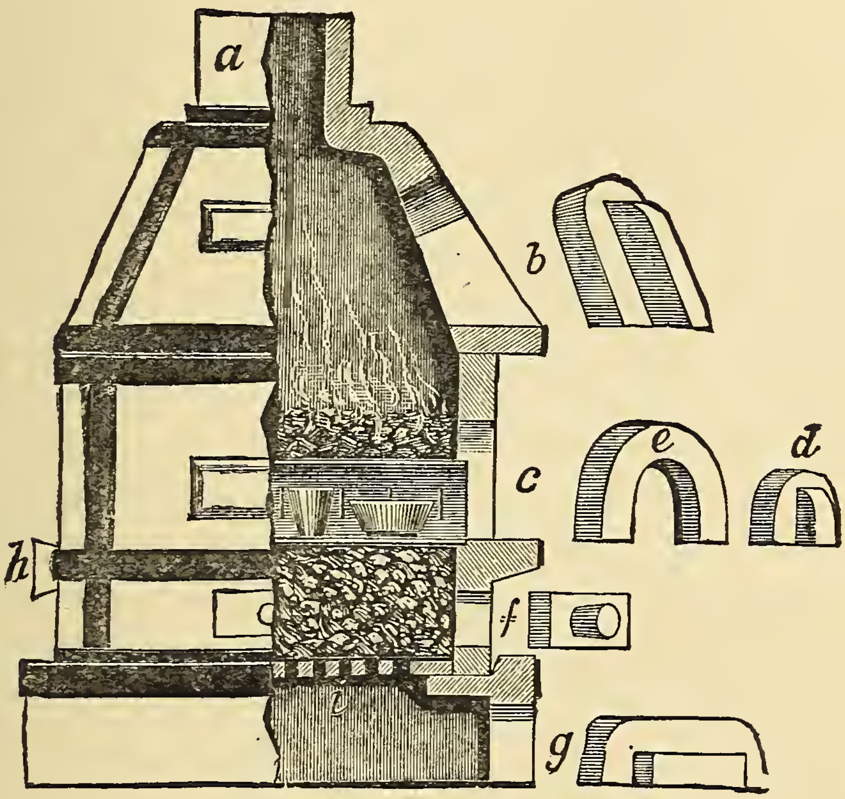
Fig. 119.—Ore Assay Melting Furnace.

Ore Assay and Melting Furnace. — A very good ore assay furnace, such as I employ in my laboratory, is manufactured at the Battersea Works, London, and is of a very simple and portable construction (Fig. 119). The furnaces are made of fire-proof material, and those of larger dimensions can be utilised for melting gold, silver, copper, and other metals.

Muffle Furnace.—This furnace (also of Battersea make) is suitable for laboratory work, and being built up in sections it can be readily packed for transportation. It is illustrated in Fig. 120.

II. Melting and Refining of Silver Bullion.

Melting Silver Bullion.—The spongy metal which has been obtained from retorting the amalgam is smelted in large plumbago crucibles, which are from 12 to 16 inches high and 8 to 10 inches in diameter on the top. These crucibles have to be annealed in a very slow fire before use, as by exposing a



a, socket on which to fix iron chimney.
b, door for fuel.
c, muffle.
d, door of muffle.
e, muffle arch.
f, door for stirring fire.
g, doors for regulating draught.
h, support for muffle.
i, iron grate.

Fig. 120.—Muffle Furnace.

new crucible to a bright fire the same will break and fly to pieces.

If the Salamander crucibles be used for the same purpose, the process of annealing is reduced to a minimum owing to their not absorbing moisture.

The melting furnaces are shown in Figs. 37 and 38 (p. 85), and Fig. 57 (p. 161). The charge for each pot varies, but usually silver ingots are cast to weigh about 80 to

100 pounds. The plumbago crucible will hold 30 to 40 lbs. of retort metal, and when this is fused two or three pieces are carefully added, and some borax, and when oxidizable metals are present some nitre is also added. In a well-constructed air furnace the whole mass will be fused in about one and a half hours, when the dross which is formed on top of the metal bath is carefully removed, and if the silver in the pot shows indications of the presence of some base metals some more nitre and borax is added, and this continued till the dross skimmings show a bright green colour.

In case the silver bullion contains lead some bone ash is poured on the metal bath, which will absorb the lead oxide, and by the addition of some borax the dross then formed can be removed. It is not advisable to continue the refining of melted silver too far, as a loss of silver by volatilization would take place.

Just before taking the plumbago pot out of the furnace, for the purpose of casting the ingot, a long iron rod is heated to redness at one end, and with it the silver is stirred thoroughly—the iron being introduced red-hot into the molten mass, no metal adheres to it. This stirring renders the metal homogeneous. A stirrer made of the same material as the plumbago crucible can be used instead of an iron rod. After the stirring the metal is sampled by using a plumbago dipper or small iron cup welded to a bent rod. This cup will hold about $\frac{1}{2}$ ounce of silver. The same is also heated to redness and dipped into the molten mass, and the contents of the cup poured into a basin of water, which granulates the silver. I generally took three to four cupfuls of silver for the assay sample, and found this method more convenient than the chipping of the ingots. The crucible is now taken up by the tongs, shown in Fig. 39, and poured into moulds. As soon as all the molten silver is in the mould some powdered charcoal is scattered over its surface, thus preventing the absorption of oxygen from the air during the cooling. The mould ought to be tolerably hot and covered with oil on the inside before casting the silver into the same.

After hardening, the ingots are placed in tubs of water slightly acidulated with sulphuric acid, and when cooled are ready to be weighed on the bullion scales. Their fineness is determined by the samples taken by the granulating process heretofore described.

The loss in melting varies from 2 to 10 per cent., according to the quantity of mercury the retort metal contains, or the percentage of oxidizable metals present.

The ingots of bullion which I produced in Idaho varied in value, as a large percentage of gold was in the ores, and one-half the value of the ingots was gold. The same was the case with the ingots from the Comstock mines. At Mineral Hill, when the ores were roasted, the bullion was an alloy of copper, lead, and silver, even after a very perfect fusion and refining; it occasionally retained traces of mercury, and had to be assayed according to Gay-Lussac's method, with the modifications for bullion containing mercury.

Melting Base Bullion at Mineral Hill.—I have mentioned (p. 166) that at Mineral Hill the working of the tailings resulted in the production of very large quantities of base bullion, which not only contained base metals like lead, copper, and some antimony, but also, and in the largest proportion, iron; such bullion was accordingly called—perhaps improperly—"iron amalgam."

The melting of the retorted amalgam was always found to give considerable trouble, as a hard, spongy metal would separate in the crucible, which refused to smelt under the strongest heat of the melting furnace. This proved to be metallic iron, resulting from the wear and tear of the battery and pans during the previous amalgamating treatment of the ores. This iron, when the ores were under treatment, did not enter into combination with the quicksilver in pan amalgamation; and it was one of the interesting observations I made, that metallic iron, when exposed in the tailing pits for some time to atmospheric influences, acquires affinity to quicksilver, through some chemical change the nature of which I am unable

to define. It is evident that the iron existed in the ore when it first underwent pan amalgamation, and that the bullion produced was comparatively free from it, as very little slag was produced in smelting the bullion produced from the roasted ore, whereas in the subsequent tailing amalgamation large quantities were taken up by the quicksilver.

Under the lens the amalgam showed some specks of metallic iron, but not sufficient to account for the very large proportion present in the retorted amalgam; and from this I was led to the conclusion that the iron was dissolved in the mercury, especially as during the separation of the lead from the silver amalgam the largest proportion of iron remained in the lead amalgam.

During the melting of the black bullion the lead would liquefy out of the metal and leave on top a hard spongy mass, and when I noticed that no more drops trickled out, I would take the sponge up with a scoop and transfer it to a crucible placed in an adjoining wind furnace, and heated beforehand to a bright redness. Into this crucible I would transfer all the skimmings and slag from two pots placed in the right and left-hand furnaces. I would now bring the middle furnace to a white heat and add sulphur, which would combine with the sponge, and after continued heating and adding of sulphur, I was able to convert the contents into a fusible mass, which was poured into conical-shaped iron pots, such as are used in lead-smelting establishments.

As the sponge contained some lead, this metal was found on the apex of the cone, and recovered after solidification of the matt. The slag on top of the cone was hammered off the matt. In this manner several thousand pounds of matt accumulated, containing about 20 to 25 per cent. of copper, and varying as to its silver contents, but on the average it realised about four shillings per pound, when sold to the smelting works.

The result of these workings showed that the base metal chlorides, produced during the chloridizing roasting of the ore, especially lead and copper, which had not been taken up

during the first amalgamation process, underwent a decomposition in the tailing pit which rendered them amalgamable; and that the small percentage of silver sulphides remaining in the tailings required a very large percentage of salt and sulphate of copper to reduce them to an amalgamable chloride. In the absence of any analytical studies on the subject it is difficult to venture any opinion as to the exact chemical changes which take place in the tailing pit, and produce results dissimilar from those obtained in the amalgamation of the original ore.

I cannot enter here into the details of the great variations in the fineness of silver bullion produced which have come under my notice while in the silver regions and in the U.S. Government Mint, but it need hardly be said that the ingots vary in every district, according to the character of the ores. At Pioche, Southern Nevada, bullion was produced 0·050 fine; and I have produced ingots at mines which assayed over 990 fine. The best bullion which came under my notice was from the Comstock mines, from the chloride ores of White Pine, Silver Reef, Utah, and from some districts of Arizona.

I append two illustrations which will be of interest. Fig. 121 shows the melting room of the Consolidated Virginia Mine, Nevada, while the casting of the ingots is taking place; and Fig. 122 represents the bullion weighing room of the Consolidated Virginia Mine, Nevada.

Assaying of Silver Bullion.—It is generally necessary when dealing with bullion whose approximate fineness is not known to make a preliminary assay. For this purpose 1,000 parts or $\frac{1}{2}$ gramme of the bullion should be weighed out and wrapped in lead foil, weighing 4 grammes; this is placed on the cupel, and after the cupellation is complete the button should be examined to see if it present all the characteristics of pure silver. If this is not the case, weigh out a fresh quantity of the silver and increase the quantity of test lead, or make

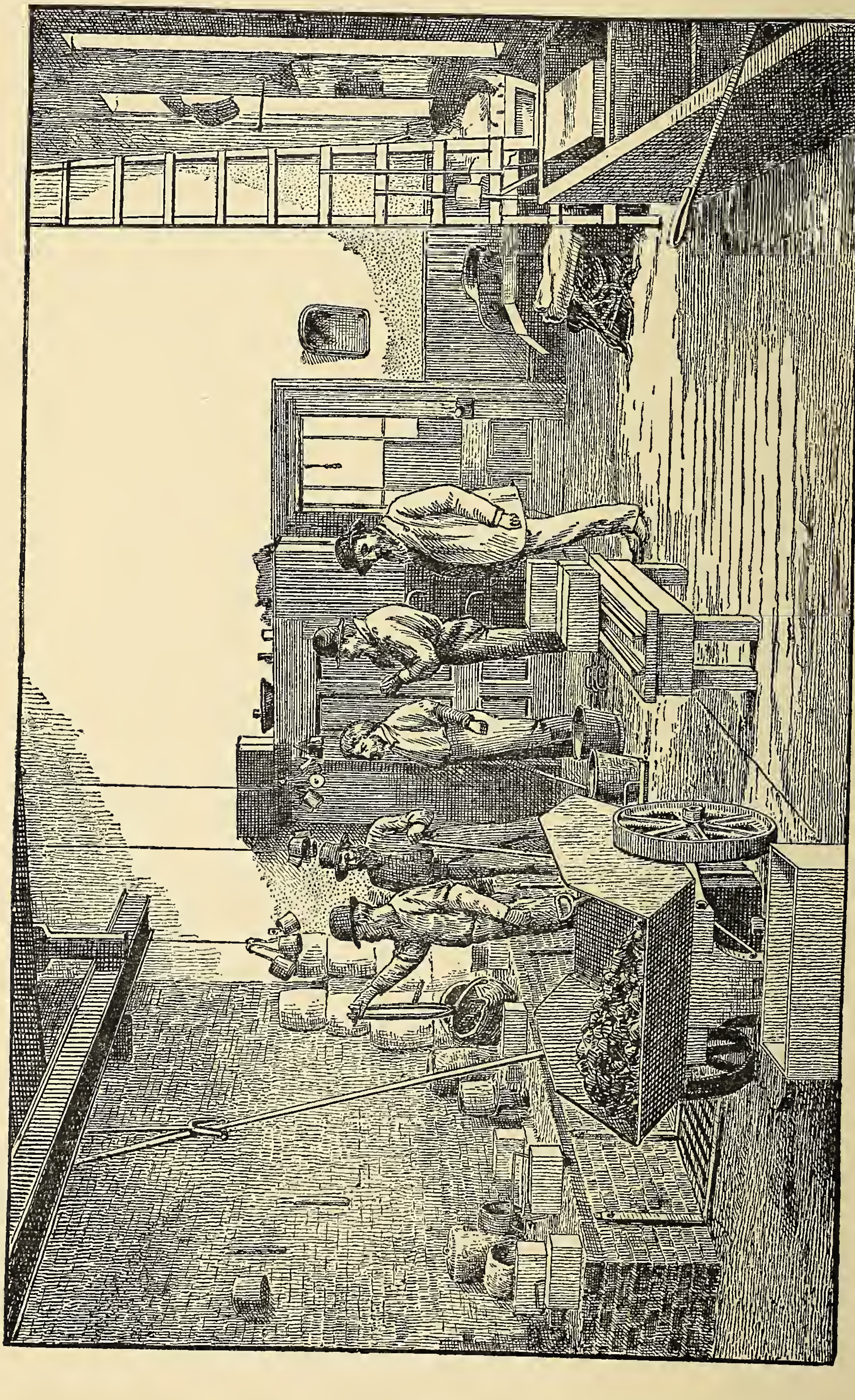


FIG. 121.—THE MELTING ROOM AT THE CONSOLIDATED VIRGINIA MINE, VIRGINIA CITY, NEVADA.

several samples at once with increased portions of test lead, and find the proper quantity of lead required. When this has been definitely ascertained, weigh out a duplicate assay of ½ gramme each of the bullion, and wrap each in the required quantity of lead foil; then weigh out ½ gramme of pure standard silver, and wrap the same in a piece of lead foil weighing 60 grains, and place the three buttons on three hot cupels, with all the necessary precautions indicated under the heading of Cupellation. When the buttons in the cupels are cold they are removed with pliers and cleaned. The two bullion buttons ought to balance one another when placed on the assay scales. They are then weighed separately; and supposing the button to weigh ·880—which means that the ingot is 880 fine, and contained 120 parts of base metals—the fine silver button is now weighed, and is found to weigh 997 instead of 1,000, which means that 3 parts of silver have been lost during cupellation; this is added to the 880, making the fineness of the bullion 883.

This correction by means of fine silver is not always to be entirely relied upon, but in far-off mining districts the fire assay is very often considered sufficient to estimate the value of ingots, especially if they are sent to the mints, but most large mining companies use the volumetric or wet method for determining the value of the silver bullion, as will be described hereafter.

The fire assay becomes the more unreliable the more base metal there is in the bullion. The presence of copper interferes also with a regular cupellation, and the proportion of copper carried off by litharge varies not only with the temperature, but even for the same temperature in relation to the amount of copper and lead the alloy contains. By cupellation of 100 parts of copper with different proportions of lead in the same furnace, Mitchell states that the following results have been obtained :—

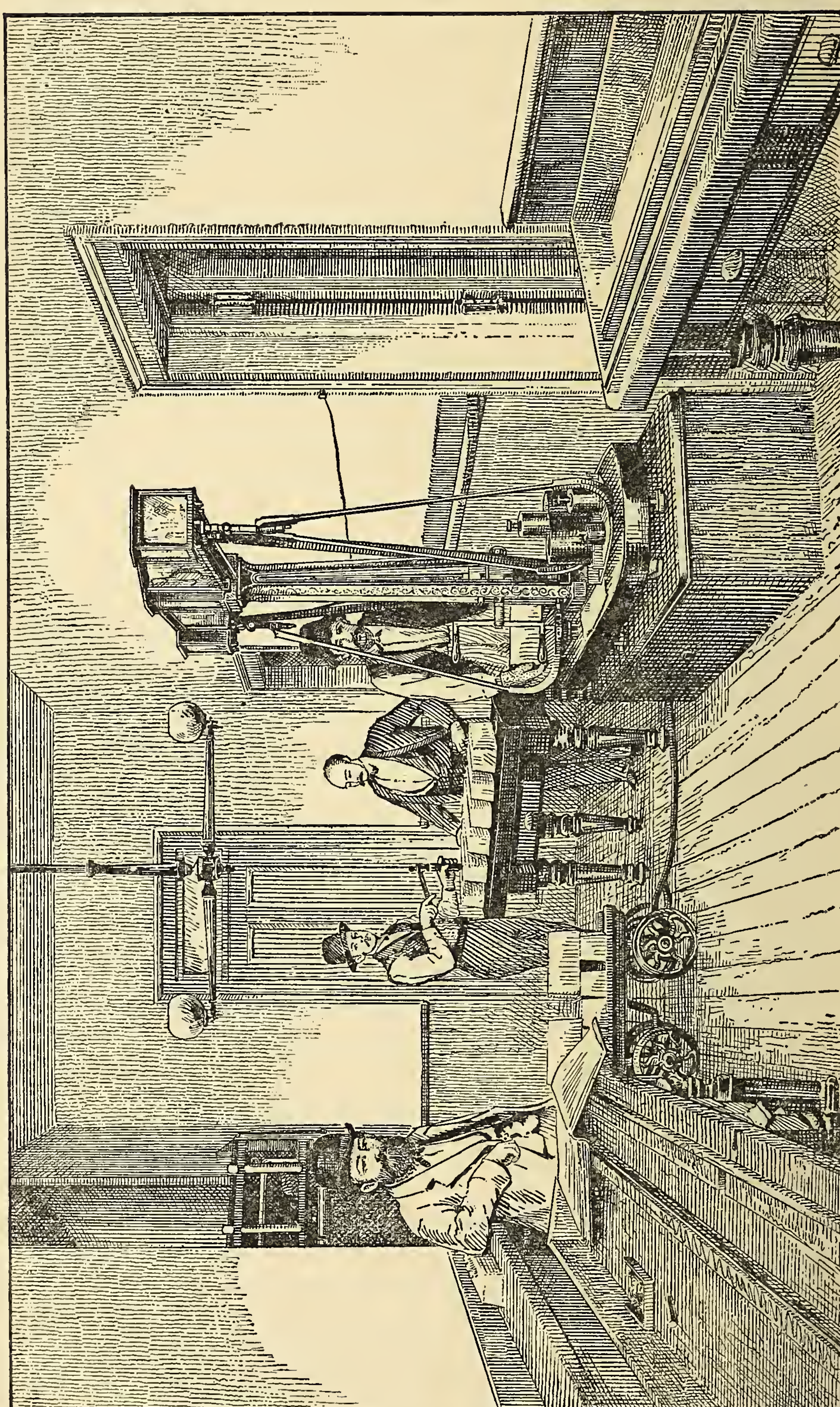


FIG. 122.—THE BULLION WEIGHING ROOM AT THE CONSOLIDATED VIRGINIA MINE, VIRGINIA CITY, NEVADA.

Lead added.	Copper remaining after Cupellation.	Quantity of Lead consumed in carrying off 1 of Copper.
100	78·75	3
200	70·12	7·1
300	60·12	7·7
400	49·40	7·9
500	38·75	8·1
600	26·25	8·15
700	19·75	8·
800	8·75	8·70
900	5·62	9·50
1,000	1·25	10·10
1,050	0·00	10·50

From this it will be seen that the lead carried away from $\frac{1}{15}$ to $\frac{1}{10}$ of its weight in copper. It is better to add the lead gradually and not at once, so as to maintain the richness of the copper on the cupel as long as possible; as, for example, an alloy of 4 copper and 1 silver will require 16 to 17 parts of lead, but only 10 parts will be needed if the lead is added gradually.

Assaying of Doré Bullion.—By "doré bullion" is understood silver containing gold. The latter having been first determined by a preliminary assay, after cupellation the assay button is weighed. Let us suppose the same weighs 922; this represents the weight of the alloy of gold and silver. On dissolving this button in nitric acid a brown powder is collected at the bottom of the flask. After decanting the acid carefully, the flask is filled with pure distilled water, and the gold powder is allowed to drop into an annealing cup according to the system described under "Assay of Gold" in my book on "Metallurgy of Gold." The cup after being heated to redness contains the bright yellow metal, which is now weighed on the assay scales. Let us suppose the same to weigh 100, which means that the ingot contains 822 parts of silver and 100 parts of gold.

The determination of the gold by this method is liable to serious errors, as, owing to the fine state of division to which the gold powder is reduced, some loss may easily occur, and

to avoid this the *synthetic assay* is made use of to determine the percentage of gold in doré bullion.

The approximate weight of the gold having been determined, it is only necessary to add to the assay pound of the bullion sufficient chemically pure gold, so that after cupellation the resulting button, after being flattened out and annealed, can be rolled into a spiral which, after boiling in nitric acid twice, will leave a solid cornet of gold behind, which can be safely manipulated.

In the case above mentioned we weigh out 1,000 parts of the doré bullion, and add to the same 200 parts of perfectly pure gold, which total weight of 1,200 parts is now wrapped in the requisite amount of lead-foil and submitted in duplicate to cupellation. The resulting buttons from the cupels should weigh 1,122 each; these are now hammered, annealed, laminated, annealed again, and rolled in rollers into spirals for the boiling in the nitric acid flasks. As the quantity of gold in the bullion was 100, the quantity of pure gold added was 200 parts. There are 300 parts of gold to 822 parts of silver in the cornet, which allows a perfect parting of the silver from the gold. The assayer should be careful to see that he has as nearly as possible three times the weight of silver to the quantity of gold present in the cornet, so as to effect a "parting" of the two metals in the boiling with nitric acid. The resulting gold cornet should weigh 300, and as 200 fine gold have to be deducted from it, it leaves 100 as the result for "gold" in the doré bullion.

Some assayers use a fixed quantity of absolutely pure gold in the synthetic assay—say, 500 parts—and add sufficient chemically pure silver to make the proportion of gold added to the doré bullion to the silver as 3 to 1. In the above case $500 + 100 = 600$ gold would require the addition of 978 of pure silver, as 822 parts are already present in the bullion, making $3 \times 600 = 1{,}800$ parts of silver. The assayer must keep perfect records of each assay quantity of gold and silver added.

III. Assaying Silver by the Wet Method.

Process of the Assay.—This method of assaying—known also as the "volumetric" method, owes its introduction to Gay-Lussac, and is based on the fact that hydrochloric acid and soluble chlorides are capable of completely precipitating silver from its solution in nitric acid without acting on any other metal with which it may be associated except mercury. The presence of this metal can be at once ascertained in the manner subsequently described, and any error which it would occasion may also be avoided.

The process may be conducted in either of the three following ways: (1) by precipitating the silver by an excess of the chloride, and calculating the weight of metal from that of the chloride produced; (2) by determining the *weight* of a standard solution of salt which is required to precipitate the silver present; (3) by determining the *volume* of a standard solution of salt which is required for that purpose.

The last of these methods is the one generally in use, and this I will proceed to describe.

The alloy, previously dissolved in nitric acid, is mixed with a standard solution of common salt, which precipitates the silver as chloride, a compound perfectly insoluble in water, and even in acids. The quantity of chloride precipitated is determined not by its weight, which would be less exact and occupy too much time, but by the volume of the standard solution of common salt necessary to exactly precipitate the silver previously dissolved in nitric acid.

The term of complete precipitation of the silver can be readily recognised by the cessation of all cloudiness when the salt solution is gradually poured into that of the nitrate of silver. One milligramme of that metal is readily detected in 150 grammes of liquid, and even a half or a quarter of a milligramme may be detected if the liquid be perfectly bright before the addition of the salt solution.

By violent agitation for a minute or two the liquid, rendered milky by the precipitation of chloride of silver, becomes suffi-

ciently bright after a few moments' repose to allow of the effect of the addition of half of a milligramme of silver to be perceptible. The presence of copper, lead, or any other metal, with the exception of mercury, in the silver solution has no sensible influence on the quantity of salt required for precipitation; in other words, the same quantity of silver, pure or alloyed, requires for its precipitation a constant quantity of the standard salt solution.

Supposing that 1 gramme of pure silver be the quantity operated on, the solution of salt required to exactly precipitate the whole of the silver ought to be of such strength that if it be measured by weight it shall weigh exactly 100 grammes, or if by measure 100 cubic centimetres. This quantity of salt solution is divided into 1,000 parts; and the standard of an alloy of silver is generally the number of one-thousandth parts of solution necessary to precipitate the silver contained in a gramme of the alloy.

Measurement of the Salt Solution.—The solution of common salt—called the *normal salt solution*—is prepared as follows: 0·5427 kilogrammes of salt and 99·4573 kilogrammes of pure water are taken to form 100 kilogrammes of solution, of which 100 grammes will exactly precipitate 1 gramme of silver; or, what is the same, 100 cubic centimetres of the solution shall precipitate 1 gramme of silver. The solution can be kept at a constant temperature, in which case the assay requires no correction; or if the temperature be variable, its influence on the assay must be corrected. These two circumstances do not change the principle of the process, but they are sufficiently important to require some changes in the apparatus, and that each of the two processes should be treated separately; one, in which the normal temperature is constantly maintained, the other in which it is variable. Experience has shown the latter to be preferable; this will be first detailed, and the other will be described hereafter.

The necessary apparatus for measuring the normal solution is very simple, consisting of a pipette capable of accurately

measuring 100 cubic centimetres of liquid; tube pipettes, graduated in cubic centimetres, and having a fine opening at the lower end; and bottles, holding about 10 oz., with carefully ground stoppers. An excellent and extremely simple form of pipette has been proposed by Stas, which renders all adjustment by the eye unnecessary, and when carefully made it gives very accurate results. It is shown, accompanied by the necessary accessories, in Fig. 123, from which it will be seen that both ends are drawn out to a fine opening, which is about 0·09 in. in diameter at the lower extremity, and 0·04 in. at the upper extremity. It is found that, if care be taken in grinding, the drop of liquid retained at the lower end is of a constant magnitude, and thus the amount delivered by the pipette is invariable.

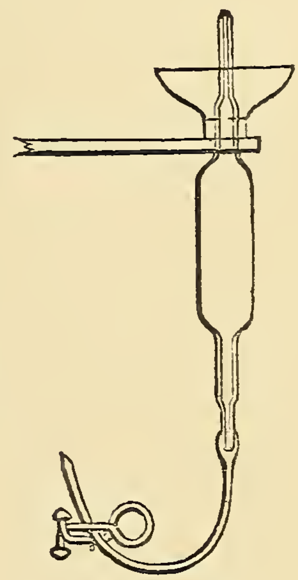


Fig. 123.—Pipette.

The salt solution is introduced at the lower extremity by means of an india-rubber tube connected with a large vessel, which is placed at an elevation above the level of the pipette. A perforated glass dish is affixed at the upper extremity to collect any liquid which may escape at the opening; and when the pipette is full the flexible tube is closed by a tap or a spring clip, the finger is placed on the upper opening, and the tube removed. When the lower portion of the pipette has been wiped by drawing the fingers over it, the bottle holding the silver solution is placed below and the upper end opened by sliding the finger off, a motion which insures the same amount being always left within the tube. This pipette holds exactly 100 cubic centimetres of normal solution.

Other methods for filling the pipette with normal salt solution will be found described in works on assaying.

Decimal solutions are prepared by taking 100 cubic centimetres of the normal salt solution and diluting the same with distilled water till it fills a flask holding exactly one litre, or 1,000 cubic centimetres. For the decimal solution of

silver 1 gramme of chemically pure silver is dissolved in a small quantity of pure nitric acid, and the liquid diluted to 1 litre.

After the preparation of the decimal solutions, several bottles must be prepared each of which contains 1 gramme of pure silver dissolved in 8 or 10 grammes of nitric acid. To these will be given the name of check or "witness" assays.

To ascertain the standard of the normal solution, pour a pipetteful into one of the check flasks, and agitate briskly until quite bright. After a few moments' repose, two-thousandths of the decimal solution of salt are added, which, by supposition, will produce a precipitate. The normal solution is consequently too weak, since the salt employed was not perfectly pure. It is again agitated and two other thousandths are added, which produce a precipitate. The addition of successive two-thousandths is thus continued until the last produce no precipitate. Suppose in all sixteen thousandths have been added, the two last which have been added are not reckoned, as they produced no precipitate; the two preceding have only been in part necessary—that is to say, the acting thousandths added are above 12 and below 14, or taking the mean, equal to 13.

Thus, in the existing state of the normal solution 1,013 parts are necessary to precipitate 1 gramme of silver, while only 1,000 should be required. The quantity of salt to be added to the normal solution may be found by noting that the quantity of salt first employed, that is to say 0·5427 kilogrammes, has only produced a standard of $1000 - 13 = 987$ thousandths, and by the following equation—

$$987 : 0{\cdot}5427 = 13 : x = 7{\cdot}13 \text{ grammes.}$$

This quantity of salt must therefore be added to the normal solution.

After having washed the tubes and pipette with the new solution, another check gramme of silver is operated on. It is found, for instance, by proceeding by one-thousandths at a

time, that the first precipitates, but the second does not. The standard of the solution is therefore too weak, being comprised between 1,000 and 1,001—that is to say, it is equal to 1,000½; this, however, is not sufficiently near.

Pour into the assay flask two thousandths of the decimal solution of silver; these will merely decompose the two thousandths of salt, and the operation will have retrograded by two thousandths—that is, it will be reduced to the point from which the thousandths were first employed. If, after brightening the liquor, half-a-thousandth of the decimal solution is added, there will necessarily be a precipitate, as was before known; but a second half-thousandth produces no cloudiness. The standard of the normal liquid is therefore between 1,000 and 1,000½, or equal to 1,000¼; this may be considered sufficiently near, but can be corrected by applying the above quotation.

The standardising of the normal solution is much less tedious than may be supposed; and it must be remarked that the liquid for a thousand assays is prepared at once, and, moreover, that in preparing a fresh solution its true standard may be very nearly obtained at once if the quantities of water and salt solution previously employed have been noted.

Temperature of the Normal Solution of Salt.—If the temperature remains constant no correction is required; but if the temperature changes, the same measure of solution will not contain the same amount of salt. Supposing the solution of salt has been standardised at 15°; if at the time an experiment is made the temperature is 18°, for instance, the solution will be found too weak, since it has become expanded and the pipette holds less than its weight. If, on the other hand, the temperature falls to 12°, the solution becomes concentrated and is found too strong. It is therefore necessary to determine the correction to be made for any variation of temperature that may occur, and works on assaying contain tables showing these corrections.

To observe the temperature of the normal solution a ther-

mometer is fixed stationary in the vessel holding the solution. Gay-Lussac has given tables of corrections for variations in the temperature. But instead of using these tables it is better to remove these variations, by adjusting the normal solution for the medium temperature of the room, and then, on each day when assaying is to be done, to first make an assay with one gramme of chemically pure silver, in order to always learn exactly the strength of the normal solution for the temperature at which the work is done. If, for example, it is found that for the precipitation of this one gramme of silver, besides the volume of 100 centimetres of the normal salt solution, one thousandth in addition of the decimal salt solution has to be added, then is this thousandth to be first deducted in all assays made at the same time, before their value is calculated. It is in practice preferable to test every time the standard solution is used for assaying the bullion in the above manner, and not to rely on correction tables. The solution is bound to vary a little by evaporation or condensation, and will fluctuate a little, and therefore its standard should be determined on one gramme of pure silver beforehand.

A dead black shelf should be fixed in front of a window (which may be glazed with yellow glass), to receive the bottles during the addition of the decimal solution, and the shelf provided with a black back of such a height that the light just passes through the upper part of the liquid contained in the bottles.

It is very essential that the solutions be kept in well-closed vessels (of glass or earthenware), to avoid evaporation; and the exact strength of the normal solution should be ascertained each morning by one or two assays of fine silver.

Pure salt is made by neutralising pure hydrochloric acid with bicarbonate of soda, evaporating the solution to dryness and fusing the dry residue, taking care to place it whilst warm in a well-stopped bottle, to preserve it perfectly free from moisture. 5·427 parts of this salt mixed with 94·573 parts of distilled water form the standard solution;

100 grains by weight of this solution precipitate exactly 10 grains of silver.

Calculating the Standard of Alloy.—By keeping the volume of the normal solution constant, it suffices to vary the weight of the alloy, taking in each particular case a weight which contains approximately 1 gramme of pure silver. Suppose the alloy has a standard of about 900 thousandths, we have the following proportion :

900 thousandths : 1000 of alloy = 1000 thousandths : x equal to 1111·1.

If that weight be now taken to ascertain the standard of the alloy, it may be found, for instance, that to the measure of 1000 thousandths of salt it is yet necessary to add 4 thousandths of salt to precipitate the whole of the silver; that is to say, that 1111·1 of alloy really contain 1,004 of silver. From this result the real standard of the alloy may be found to be 903·6, by the following equation :—

1111·1 : 1004 = 1000 : x equal to 903·6.

We may here take two examples of assaying pure or nearly pure silver, the temperature of the normal solution of salt being that at which it was standardised.

First Example.—Let the silver ingot under the assay have an approximate fineness of 990, previously determined by fire assay. 1 gramme of it is dissolved in 10 grammes of nitric acid in the assay bottle. Then pour into the bottle an exact measure of 100 c.c. of the normal solution of salt, and shake the bottle well so as to brighten the solution. As we know that a surplus of salt solution has been employed, this surplus has to be neutralised by the decimal solution of nitrate of silver.

Ten thousandths of this latter solution is poured into the bottle; it becomes cloudy, and is well agitated. On adding an extra thousandth no cloudiness is produced, so we neutralise this one by adding 1 c.c. of the decimal salt solution. We add another c.c. of the decimal salt solution, which pro-

duces cloudiness, then another, and at last a third ; this last one gives no precipitate, so was useless; the second one was only partially required : therefore the silver was $990 + 1\frac{1}{2} = 991\frac{1}{2}$ fine. In marking the addition of the decimal solutions on a blackboard, the plus (+) sign is used for salt, and the minus (—) sign for silver solution.

Second Example.—Supposing that the ingot has a presumed standard of 895 thousandths, and the temperature of the normal solution is invariable—

$$895 : 1{,}000 = 1000 : x \text{ equal to } 1{,}117$$

which is the weight of the alloy to be taken, after dissolving in nitric acid and the addition of the standard solution—and that the first 3 c.c. of decimal nitrate of silver solution give a cloudiness but the fourth not. Therefore $2\frac{1}{2}$ c.c. were required, so that the fineness of the ingot was 895 less $2\frac{1}{2}$, or $892\frac{1}{2}$.

Reduction of Chloride of Silver.—Chloride of silver can be reduced without sensible loss, after having been well washed, by plunging into it scraps of iron or zinc, and adding dilute sulphuric acid in sufficient quantity to set up a slight disengagement of hydrogen gas. The whole can be left to itself, and in the course of a few days the silver is completely reduced. This point can be easily determined by the colour and nature of the product, but better still by treating a small quantity by ammonia, which, if the chloride is perfectly reduced, will give no precipitate or cloudiness on treatment with an acid. The chlorine remains in solution in the water combined with zinc or iron. The residue must now be washed. The first washings are made with acidulated water, to dissolve oxide of iron which might have formed, and the following with ordinary water. After having completed the washing, as much water as may be left is decanted, the mass dried, and a little powdered borax added. Nothing now remains but to fuse it. The powdered silver being voluminous, it is placed by separate portions into the plumbago or salamander crucible, in proportion as it sinks.

The heat should be at first moderate, but towards the end of the operation should be sufficiently high to reduce the silver and slag to a state of complete liquidity. If it be found that not quite all the chloride was reduced by the iron or zinc, a little carbonate of potash or soda may be added to the powdered silver. The standard of silver thus obtained is from 999 to 1,000 fine.

Assay of Silver Alloys containing Mercury.—Whenever mercury is present in solution with silver under assay, it is thrown down as insoluble chloride, and the result is that the assay is rendered inaccurate. The presence of mercury in silver can be readily detected by the remarkable change which occurs in chloride of silver on exposure to light when free from mercury; but if the smallest quantity of the latter metal be present, no purple colouration of the precipitated chloride of silver will take place. This source of error was removed by M. Levol in the following manner. The sample being dissolved, as usual, in nitric acid, it was supersaturated with 25 cubic centimetres of caustic ammonia; the pipetteful of normal solution was then added, and the excess of ammonia supersaturated with 20 cubic centimetres of acetic acid, and the operation continued in the usual way.

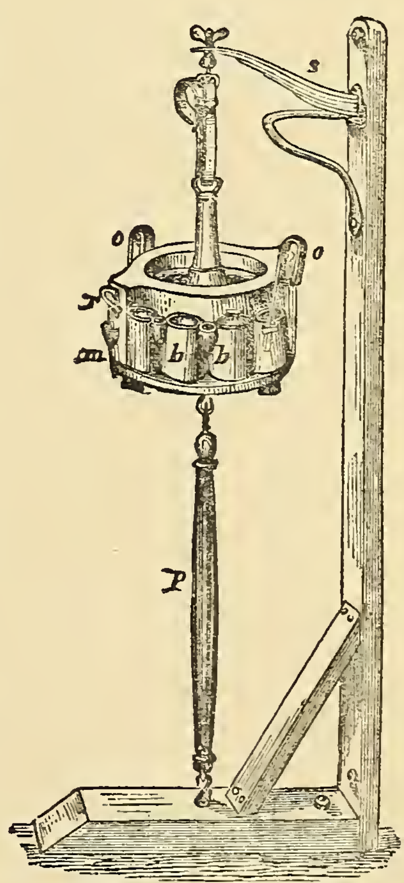


FIG. 124.—THE AGITATOR.

The Agitator.—For shaking the bottles after the salt solution has been introduced, an apparatus called an "agitator" is used (Fig. 124). This is a convenient appliance for shaking ten bottles at a time, and much saving of labour attends the use of it where a considerable number of assays have frequently to be made. It consists of a metal

frame, *m*, divided into compartments, *b b b*, to receive the bottles, and a lid, provided with spring slides, prevents the bottles from being displaced and the stoppers from being shaken out. The frame hangs from a short carriage spring, *s*, firmly fixed to the wall by a leather strap or by india-rubber door springs, and it is connected with the floor by a long india-rubber spring, *p*.

CHAPTER XV.

SUPPLEMENTARY.

THE M. P. BOSS PLANT—The Boss One-Level System—Boss Amalgamating Pan—Brunton's Ore Sampler—Roasting with Salt—What Roasting Furnace to adopt—The W. H. H. Bowers Furnace—The R. L. Thompson Furnace—Comportment of Other Metals in the Amalgamation of Silver Ores.

The M. P. Boss Plant.—Since the foregoing chapters were written I have received information that the Boss process (see *ante*, p. 133) is now successfully at work at various mines in the United States. At the Jay Gould Mine, in Montana, it is used for amalgamating gold ores, and 95 per cent. of the gold (I am informed) has been extracted there by its means. Subjoined is a description of the plant, as shown in Fig. 125.

The apparatus is arranged on benches. On the upper bench are two or more hoppers, A A′, which are connected together so that the pulp which is admitted to one will overflow into the other. The pulp is introduced into the hopper, A, through a trough, B, and when this hopper is filled to the height of the dividing-partition it overflows into the hopper, A′. This latter hopper has also an overflow through the holes, *a a*, in its side, which are at about the same height as the top of the dividing-partition. The discharge or outlet through the lower end of each hopper is regulated by a cone-shaped plug formed on the end of a screw rod, C, which passes through a cross bar, D, and has a hand-wheel, E, at its top, so that by elevating the plug the size of the hole is increased. A trough, F, passes along below the discharge passage of the hoppers, and extends out to one side, where two inclined spouts G, G′, lead from it. A conveyer-screw, H, is

placed in this trough, and is rotated by power applied to a pulley on its end, so that the pulp is fed along by the screw.

On the next bench below, a number of pans, I I I, are placed side by side, the two adjoining pans at the right hand end of the series being placed so as to receive the pulp from the spouts G G′. These pans are all placed close together, and

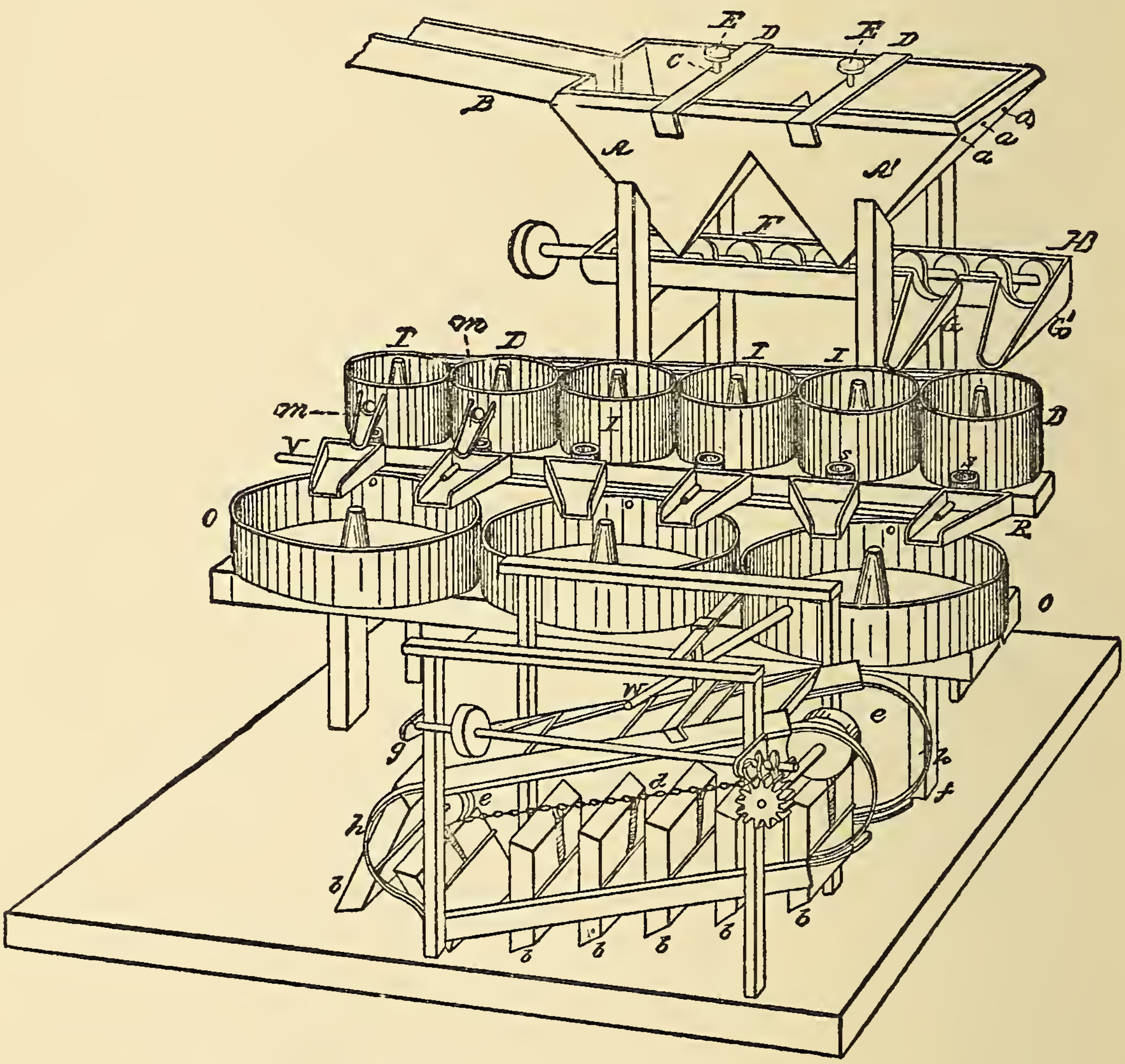


FIG. 125.—M. P. BOSS APPARATUS FOR AMALGAMATING ORES. Perspective View.

communication is established from one to another by means of openings, *j*, near their upper edges, so that they will overflow into each other, see Fig. 126.

An inclined trough, K, extends along outside the entire series, near the top of the pans, and each pan has an opening, *l*, leading from it into the trough at about the same level as the

openings, *j*. The two adjoining pans at the left-hand end of the series have each an overflow opening and spout, *m*, which discharges the pulp into a lower series of pans, hereinafter described.

It will be noticed that each pan has two overflow openings,

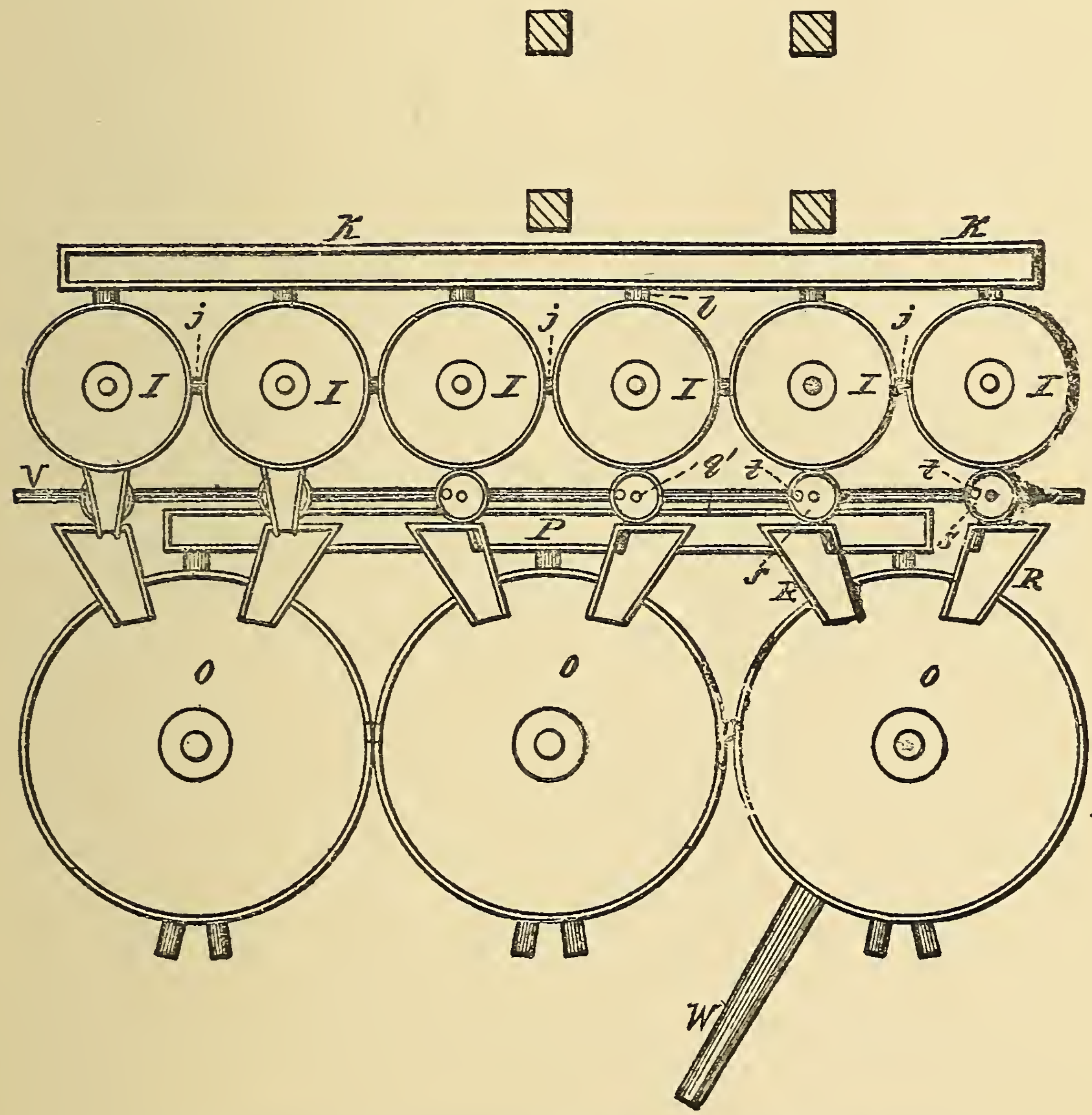


FIG. 126.—M. P. BOSS APPARATUS FOR AMALGAMATING ORES.
Top View of Pans and Troughs.

one connecting it with the adjoining pan and the other with the trough, K, which connects the entire series; so that by plugging up the openings in either pan it is cut out of the circulation without interfering with the operation of the remaining pans. The object of having two receiving and two discharge

pans at each end of the series is to enable the feed to be shifted into one or the other, as desired, in case one should become clogged or require repairs. A gate (not shown) will therefore be placed, across the trough, K, it is desired to close.

On the next lower bench one or more larger pans, O O O, are placed. These pans are connected so as to overflow into each other in the same manner that the upper series of pans are

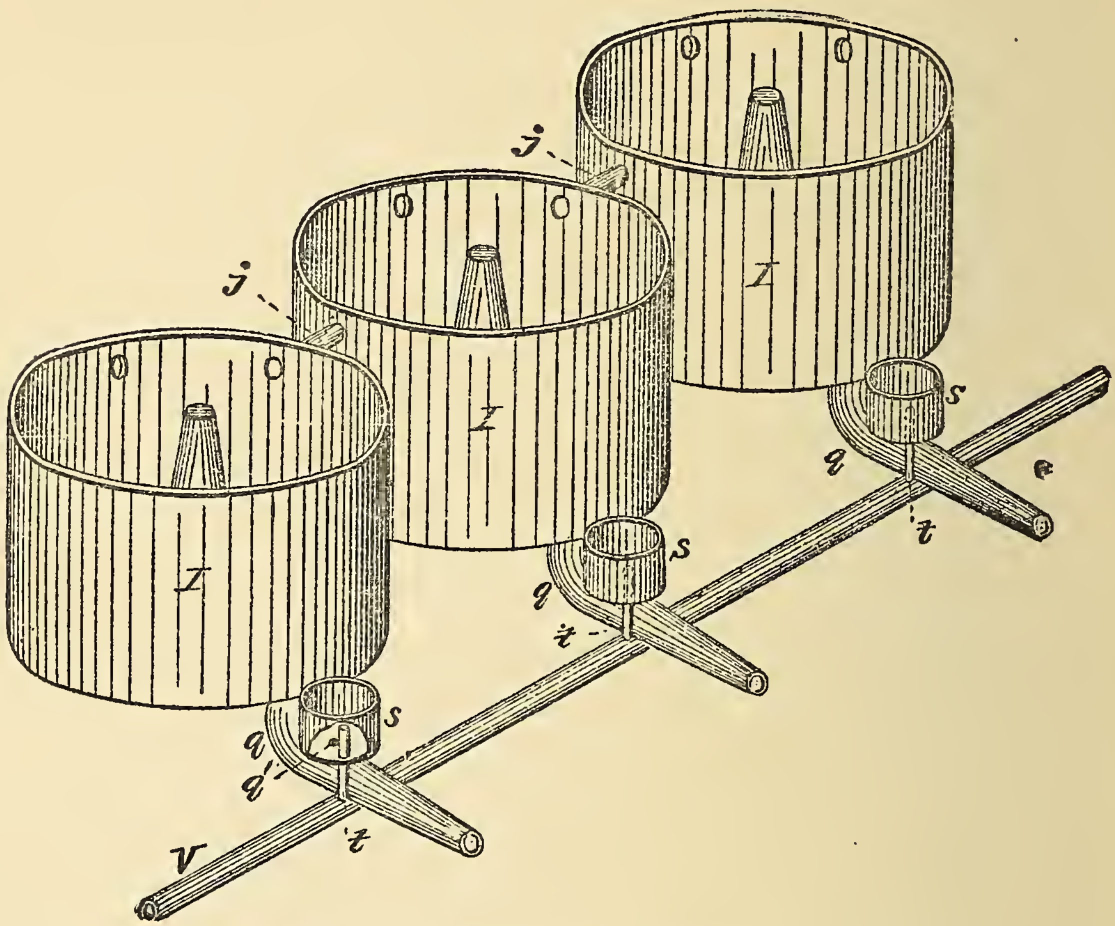


FIG. 127.—M. P. BOSS APPARATUS FOR AMALGAMATING ORES.
Pans and Quicksilver Cups.

connected, and a trough, P, connects their side overflows in the same way. Each pan, I, has a discharge-tube, *q*, Fig. 127, connecting with its bottom, which leads into a spout, R, and the spouts of each two adjoining pans, I, empty into the same pan, O, as shown. A quicksilver cup, *s*, is placed on each discharge-tube, *q*, into which the quicksilver will rise with the level of that in the pan through the orifice, *q′*, when the end of the tube is plugged. Inside of each cup is a short tube, *t*,

Figs. 126 and 127, the lower end of which connects with the main tube, *v*, which extends along under the entire number of cups. The upper end of the tube, *t*, does not extend to the top of the cup, so that the quicksilver will overflow into the upper end of the tube, and be conducted into the main pipe, *v*. This main pipe connects with a quicksilver pump, by which the quicksilver is raised and introduced again into the pans.*

The *modus operandi* of the process is as follows:—The pulp being first introduced into the hopper, A, the heavier part settles, and the lighter part overflows into the hopper, A^1, the lighter part of the latter portion running off through the holes, *a a*. The cone plug being properly regulated, the pulp falls into the trough, F, and is conveyed by the screw, H, to the spout, G, down which it passes into the pan, I. It then overflows from pan to pan until all are filled, the current being strong enough to prevent it from settling. In one or more of these pans a constant stream of mercury is supplied, which filters through the pulp, amalgamating the metals, and passing down rises in the quicksilver cup until it overflows into the circulating pipe; thence it is drawn by a pump in the usual way and introduced into the pan or pans again, thus giving a continuous circulation of mercury through the pulp and keeping the mercury in good condition. The quicksilver used in the process will sink on all occasions, by reason of its great relative weight, and will find its way into the pipe *v* ultimately, where it is submitted to the action of the pump, or is otherwise elevated and restored to the pans containing the pulp.

If desired, one or more of the pans can be cut out of the circuit after they are filled by plugging the passages and the pulp contained in them treated chemically. The pans, I, and settlers, O, are all of them of the kinds in ordinary use.

Each series of pans and settlers stands upon nearly or quite the same level, and each runs full or nearly full of pulp, so as to furnish time and opportunity for the precious metals to amalgamate with the quicksilver and separate from the worth-

* A description of the quicksilver pump which is mentioned above will be found at page 210.

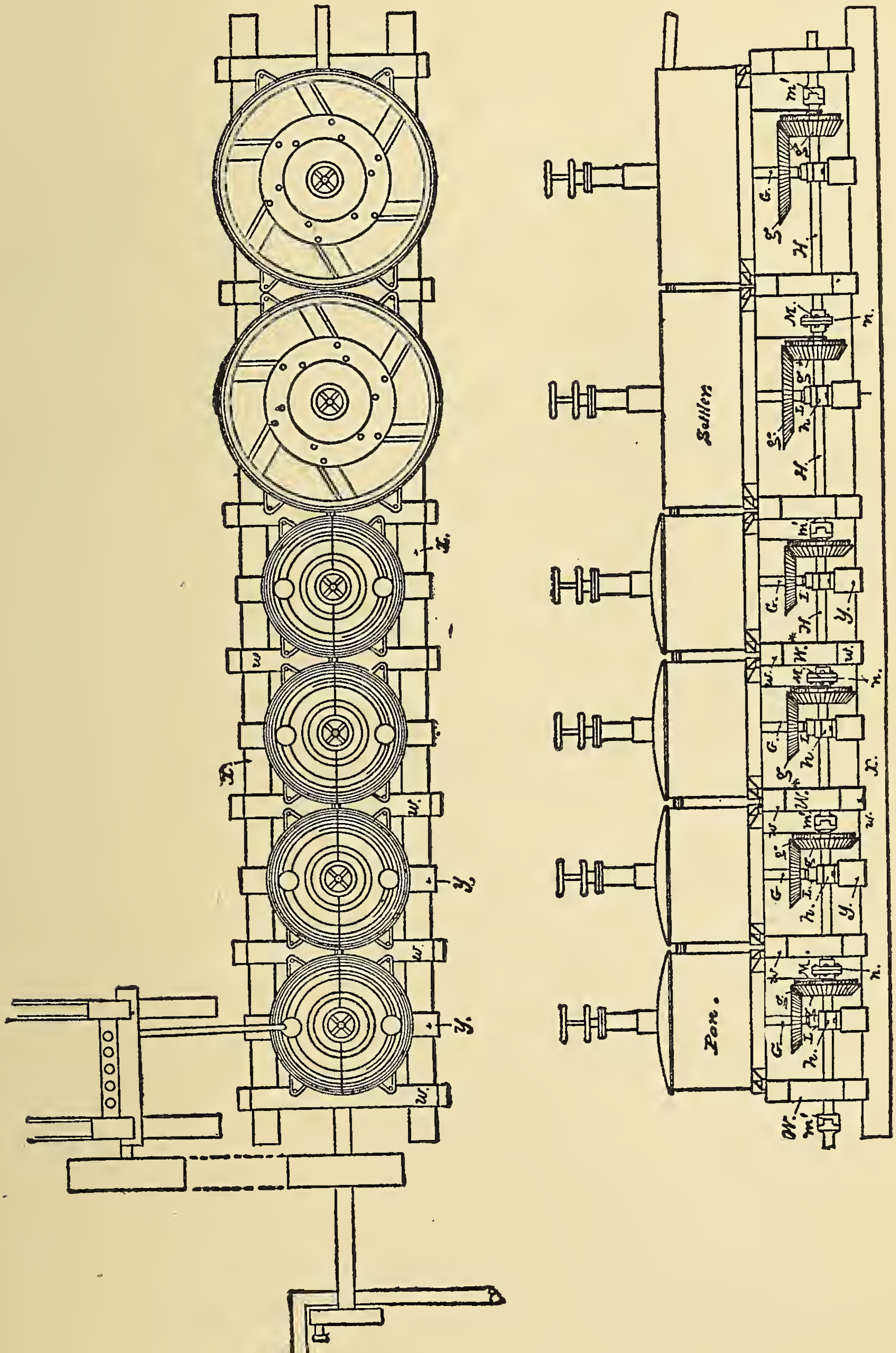
less mass of the pulp. By this apparatus the pulp is constantly fed into the first one or two pans, and is constantly passing through the series of pans and settlers, being reduced as it goes along, and the vast mass of pulp is constantly discharged from the last charge pan or settler, O.

The Boss One-Level System.—This method is certainly very advantageous, as Mr. Boss arranges his whole series of amalgamating pans in line and close connection, and also his series of settlers on the same level and connected together to discharge by overflow from one series into the other. In this way both the cost of construction and the labour of running the apparatus are reduced. He also drives the muller shafts of the whole set of pans and settlers direct from the engine shaft.

To erect the apparatus he proceeds as follows:—Longitudinal timbers or sills, X X, upon a suitable bench or level form a solid base for a number of framed supports (*see* Figs. 128, 129). Upon the sills are transverse timbers, *y y*, for shaft boxes, *h h*. The framed supports consist of two horizontal timbers, *w*, at top and bottom, and upright timbers or legs, *w**, that are framed into the horizontals and set with a slight rake or inclination outward. The top timbers receive the bottoms of two adjacent pans which are fastened down to the supports by flanges and bolts. The upright shaft, G, of each pan is carried at the lower end in a step bearing, L, provided on the top of the shaft-box, and its connection with the driving shaft is made by a mitre-gear, *g*, and a driver *g**, on the horizontal shaft. These drivers are not rigid on the shaft, but each one is connected by a friction clutch of such character that it locks the gear to the shaft under ordinary strain and conditions of work; but with any increase of resistance to the movement of the muller, and as this resistance becomes too great, the clutch yields to the strain and relieves both the driving and the driven shaft. The application of these friction clutches permits the connection of the muller and the driving shaft to be made with gearing.

The driving shaft, H, is made in sections, each one of which

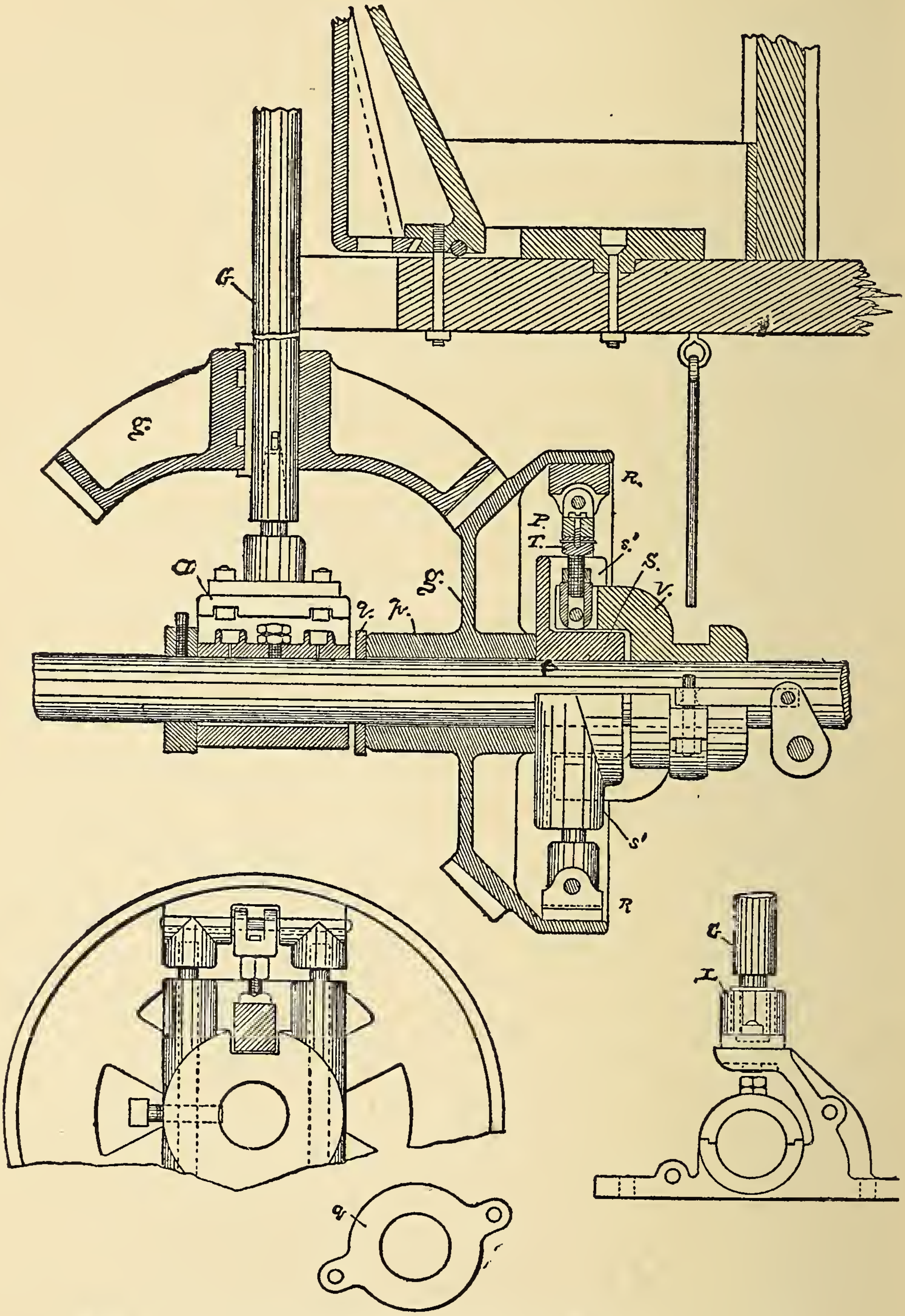
carries the gear and connected mechanism for one pan, so that



FIGS. 128, 129.—THE BOSS ONE-LEVEL SYSTEM.

the driving mechanism of any one pan may be removed for

repairs by detaching and taking out the proper section of shaft,



FIGS. 130, 131, 132.—THE BOSS PAN-DRIVING GEAR AND CLUTCHES.

without disturbing the connections of the other pans with the

shaft. The construction of driving-gear and clutches on a section of shaft is shown in detail in Figs. 130, 131, 132.

The driving gear, g^*, has a long hub, *p*, and is placed loosely on the shaft. It has a concentric rim that projects from the base of its bevelled face, and is of sufficient width to give an internal surface for two friction blocks, R R, to set against. Upon the shaft a box, S, furnishes guides, *s′ s′*, for two toggles, T T, that are attached at their outer ends to the frictional blocks R. The inner ends of the toggles are attached by joints to the arms or extensions by a sliding hub, V. This hub turns with the shaft, but is free to slide longitudinally, and the toggles, T, are attached to it by hinge joints. A longitudinal movement of the part V in one direction draws the blocks, T, away from the rim, and in the other direction sets them against the rim and locks the gear. The rods, T, are extensible, as shown, so that the amount of friction between the surfaces of the clutch can be regulated to any required degree of resistance.

The shaft sections are of equal length, and there is one section to each pan. One end of the shaft has the half of an ordinary flange coupling, M, to join the corresponding end of the next section, and the opposite end of the shaft has the one-half or part of a clutch-coupling *m′*. When the sections are set in line, the flanged ends of every two sections are united by bolts in the usual manner. This leaves the clutch ends *m′* of every two sections facing each other, and the line of shaft is completed by joining them together. Between the two flanges, at the time of coupling, a ring or collar, *n*, is inserted, the thickness of which is equal to the amount of longitudinal movement required to separate the two parts of the clutch-coupling. To remove a section of shaft, the flange-coupling is first separated, and the ring, *n*, is then drawn out to permit the shaft to slide back.

The operation of taking out any one of the shaft sections and the connected mechanism is thus very simple, as will be readily understood from the illustrations and description here given.

Boss Amalgamating Pan.—In addition to the process above described, Mr. Boss has also introduced an improved pan, which should be described here. It does not differ materially from the pans heretofore described except in the construction of the steam chamber, with the view of providing the greatest possible heating surface.

Fig. 133 is a vertical section of the pan, and Fig. 134 a bottom view, a section only of the outer portion of the pan being shown.

A are the bed timbers; B is the pan supported thereon, and having the cone centre *b*; C are the sides; D, the mullers; E, the arms of the mullers; F, the rotating head to which the arms are secured; G, the shaft for driving the head, and H the adjusting screws.

In order to provide a steam chamber or space of as great an extent as possible to obtain a large heating surface, the plate, piece, or casting, I, is formed with an annular plane surface, *i*, a cone centre, i', and a tubular extension, j^2. This is fitted to the under side of the pan, as shown in Fig. 133, its cone extending upwardly within the cone, *b*, of the pan, and its tubular extension projecting through and beyond said cone into the head, F, and forming the bearing for the shaft, G. A space, J, is left between the piece, I, and the bottom and cone of the pan, forming the steam chamber, a packing being placed at *j* in an annular rib, j', under the pan, and a rust or other steam-tight joint being formed at j^2, between the top of the cone, *b*, and the base of the tubular extension, i^2. The piece I is tightened to its seat by screws, *k*. L L are the steam ports communicating with the chamber, J. It will thus be seen that both the bottom and the cone of the pan are adapted to be heated.

In addition to forming the seat for the packing, the rib, j', in connection with an inner and concentric rib, j^3, on the bottom of the pan, serves another purpose. When the shoes and dies are not fitted accurately in the same vertical plane, it is not unusual for the edges of the former to be worn into a point, and those of the latter to be bevelled or rounded down. This brings the shoes into contact with the bottom of the pan, which being of

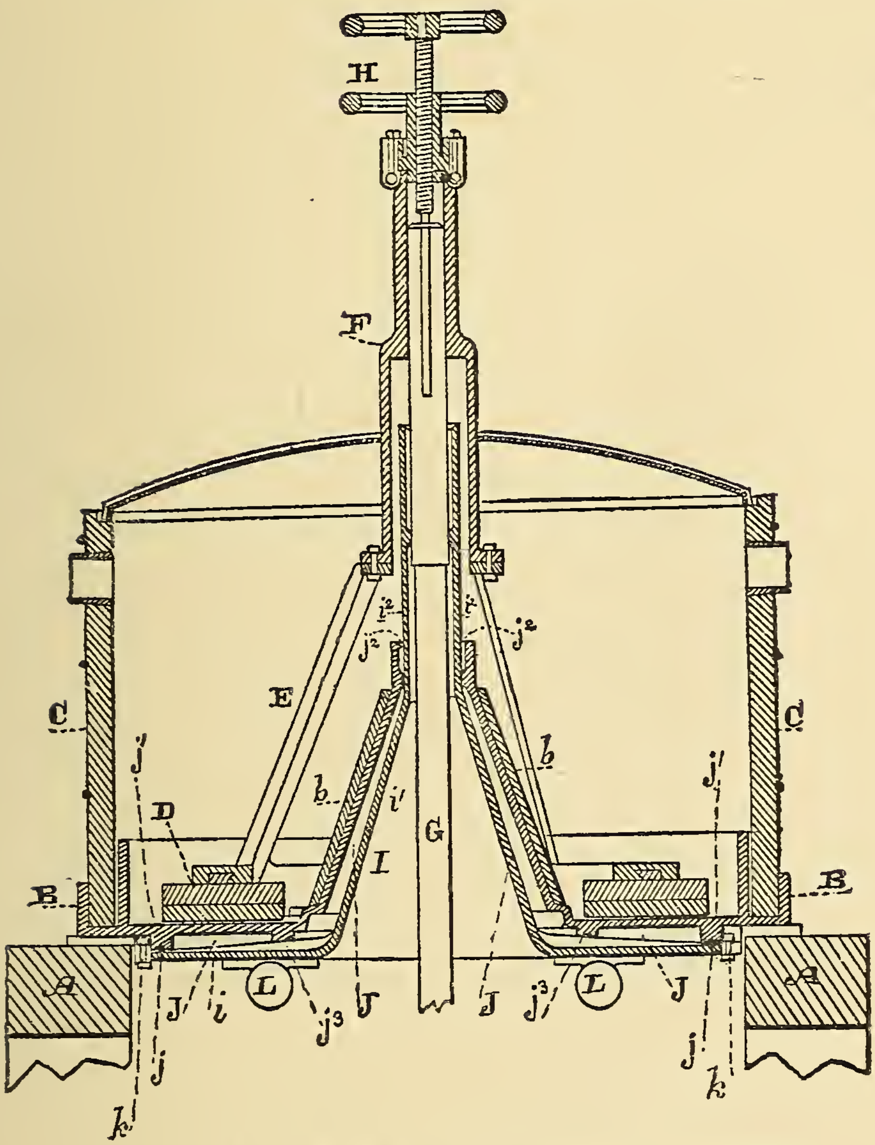


FIG. 133.—M. P. BOSS AMALGAMATING PAN. Section.

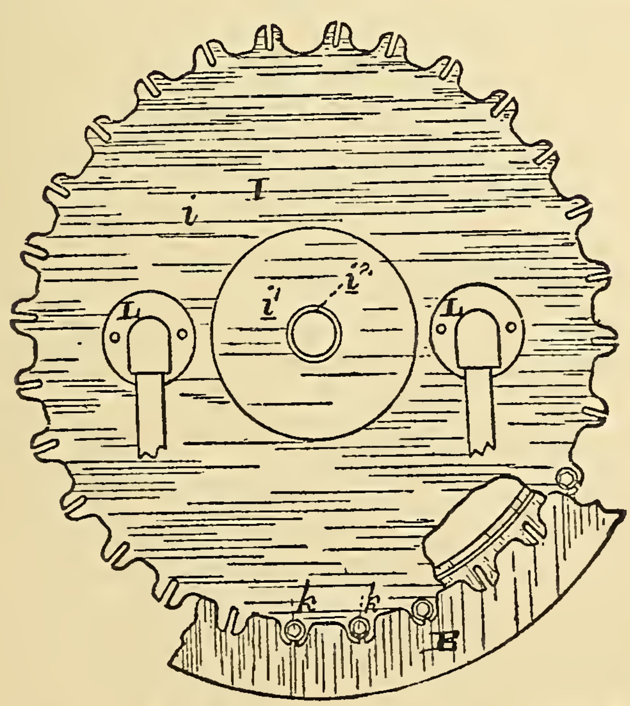


FIG. 134.—M. P. BOSS AMALGAMATING PAN. Bottom View.

much softer iron soon wears through, but by having these ribs at this point of wear the pan cannot be worn through.

The function of the pan is twofold—the pan being intended first to grind and then to amalgamate. The trituration of the ore particles in a stamping battery is never carried far enough to reduce them to the degree of fineness which will adapt them for pan amalgamation ; and the pulp is exposed, therefore, to a grinding operation in the pans, which operation brightens any metallic particles in the ore, and reduces it to such a fine state of division that in the subsequent period of amalgamation the quicksilver will act the more readily. With the more modern grinding machinery—such as Globe mill, for instance—there would be economy in reducing the pulp at once to the required degree of fineness, especially in dry crushing, as in wet crushing too much slime would be produced by too fine crushing.

The ore is generally ground for two hours ; then the muller is raised and quicksilver introduced, when through the active agitation it is divided into small globules, distributed through the mass of pulp ; these come into contact with every particle of metal, and the current in the pan is such as to throw the pulp to the periphery, leaving a cone in the centre down which the pulp slides to centre through the openings of the driver under the shoes, where it undergoes intimate contact with the mercury and escapes out at the periphery, to follow this evolution continually. To assist the current being thrown between the muller and bottom of dies, there are wings fixed on the inside of the pan, and I have used two wings suspended from two cross-pieces.

The loss of iron in the pan is considerable, not only through the grinding, but also from chemical action, and ranges as high as five to eight pounds per ton of ore treated.

Brunton's Ore Sampler.—There are many contrivances in existence whereby the sampling of ore is done automatically by machinery. They are generally so arranged as to take a certain proportion of ore, and where ores are purchased by custom the question of sampling is a most important one. The

samplers are mostly arranged by dividing or cutting-out from a falling stream of ore—by means of narrow spouts, dividing flanges, or travelling buckets—a certain percentage, which is taken to represent an average sample of the whole.

Mr Brunton has devised an apparatus which does away with the objections which can be raised against some other contrivances for the same object, as by his method the sampling of the ore can be carried out with extreme rapidity and at the

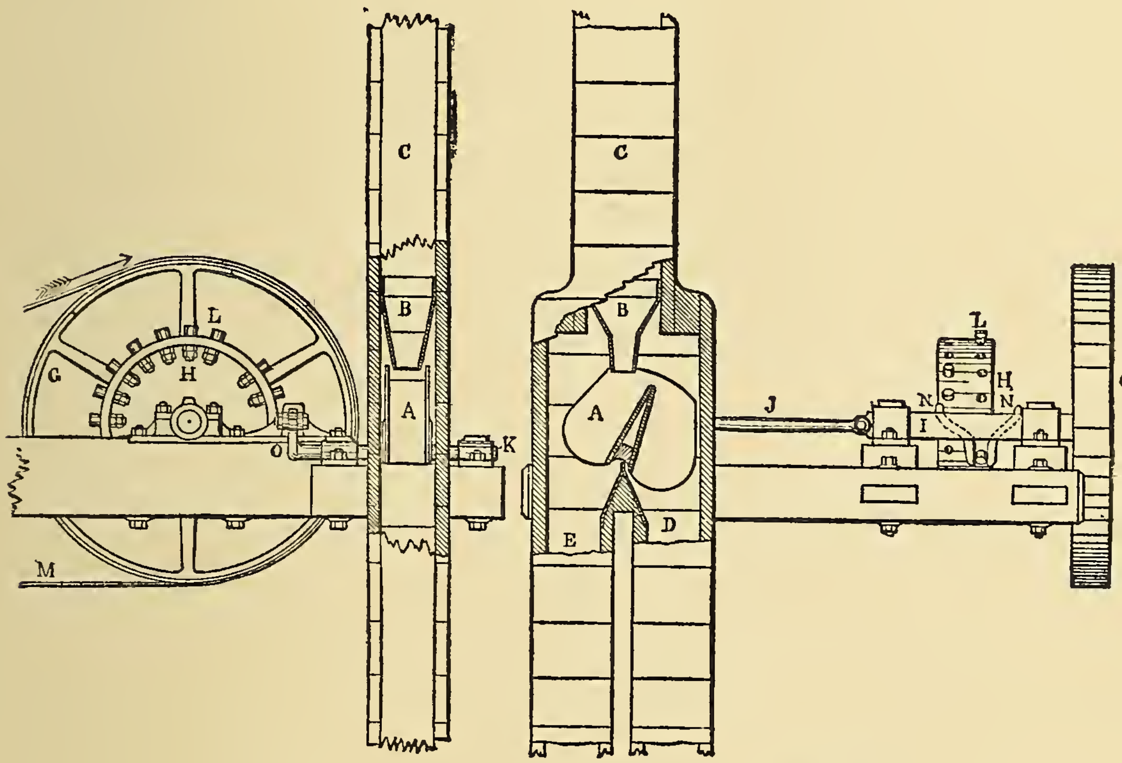


FIG. 135. FIG. 136.

BRUNTON'S ORE SAMPLING MACHINE.

lowest possible expense for power, labour, and repairs. The principle on which his sampler works is the deflection of the entire stream of ore, alternately to the right and left, into two separate portions, the relative proportions of these two divisions to each other being determined by the difference in time between the deflections to the right and the deflections to the left, as shown in the illustrations (Figs. 135, 136).

C is a vertical or inclined spout containing the falling stream of ore. B is a funnel for narrowing the width of the falling stream, so as to reduce to a minimum the necessary travel of

the deflecting chute, A. This chute, A, is pivoted upon the rock-shaft, K, and when it is deflected to the right the entire stream of ore is thrown into E, and when it is deflected to the left the entire stream is thrown into D. The deflection is caused by the movement of the crank, O, receiving its motion from the driving-bar, I, and connected with it by the pitman, J. The driving-bar, I, receives its motion from the pins, L, in the face of the revolving wheel, H, which is driven by the pulley, G, receiving motion from the belt, M, or any other suitable driving device. The face of the wheel, H, is perforated by two rows of holes, the distance between the two rows being the same as the necessary movement of the crank, O. Into these holes are inserted a number of pins, L, held in place by jam nuts on the interior of the wheel face. Preferably, twenty holes are bored in each row, and eighteen pins are employed, each hole or pin representing five per cent. of the time necessary to complete a revolution of the wheel. Now, if fifty per cent. of the pins are placed in the right-hand row of holes and fifty per cent. in the left, then the revolution of the wheel, H, carrying the pins, L, through the guides, N N, on the driving-bar, I, will hold the deflecting chute, A, on the right during one half of the revolution, and on the left during the other half, thus dividing the stream into two equal portions. If twenty per cent. of pins are placed in the right-hand row and eighty per cent. in the left, then the deflecting chute, A, will be held on the right during one-fifth of a revolution, and on the left during four-fifths, thus throwing twenty per cent. of the ore into the spout E and eighty per cent. into the spout D, &c.

In practice the wheel is driven with considerable velocity, so as to make the divisions of time as minute as possible without injuring the machine by excessive vibration.

It is evident that the pins, L, can be placed on the face of wheel, H, in two rows, and the guides, N N, attached directly to the pitman without the intervention of drive-bar, I.

Roasting with Salt.—In some cases it is advisable to roast the ore until all the sulphurets present are decomposed and reduced to the form of oxides and sulphates; then add

salt, and subject to a chloridizing roasting. If salt is added at once to the raw ore, a large portion of salt is consumed to convert the base metals into chlorides. This arises from the fact that when sulphurets of iron, copper, lead, zinc, and the like, are exposed to the action of chlorine gas or volatile chlorides at a moderately high temperature, they are decomposed with the formation of chloride of sulphur and chloride of the base metals wherewith the sulphur was combined. This reaction takes place at a lower temperature than that required to oxidize the sulphurets; hence it is apparent that if cold raw ore be introduced into a furnace more or less filled with chlorine, a large proportion of the sulphurets present will be converted into chlorides.

When, on the other hand, oxides of iron, copper, lead, zinc, and the like are exposed to the action of chlorine gas at high temperatures, the only effect produced is to convert the lower oxides into such higher ones as can subsist at the temperature to which they are exposed. Thus, for example, protoxide of iron would be converted into sesquioxide; suboxide of copper would be converted into oxide. Oxide of lead would probably be unaltered; the same would also be the case with zinc. It is evident, therefore, that if a roasted ore be mixed with salt and again roasted, a much smaller quantity of the base metals will be converted into chlorides than if the raw ore had been mixed with salt and roasted.

When a roasted ore containing base chlorides, and notably chlorides of lead or copper, is amalgamated, these chlorides are reduced by the iron of the amalgamating pans, and the metallic copper and lead thus formed amalgamated; hence the presence of base chlorides in a pulp which is to be amalgamated is disadvantageous, because they produce a greater corrosion of the pans and an impure and dirty amalgam, which when retorted yields an impure and base bullion.

What Roasting Furnace to adopt.—This is a point which can only be decided in a particular case by practical experience. Some ores will admit of a very advantageous

roasting in revolving cylindrical furnaces, especially if the ore will not "ball." There is much loss of silver by volatilization. The reverberatory furnaces are to be preferred, as they admit of a more proper regulation of the draught and heat, and the salt can also be added at the proper moment. Among reverberatories the furnace which is most economical in use is the three or four-hearth continuous furnace, which requires the least fuel, as one cord of wood will roast as much as six tons of ore.

The furnaces used in the metallurgical treatment of silver ores are (1) the reverberatories for hand stirring; (2) the reverberatories with stationary hearth and automatic stirring apparatus; (3) the automatic reverberatory furnace with revolving hearth; (4) revolving cylinders; (5) shaft furnaces of the Stetefeldt type.

It will be well to give here a sketch of the roasting operations in the long reverberatories, which in some localities have been found for certain silver ores to give better results than any other furnaces, and offer economically also many advantages.

The single-hearth furnace has been described at page 149, and the long reverberatory only differs from that furnace in that the hearth is either continuous or separated by steps a few inches in height. Such a hearth cannot be built of indefinite length, and it must be apportionate to the quantity of sulphur the ore contains, as in a long furnace the heat from the fireplace is carried only to a certain distance in the furnace, and will at a certain point be so cooled down as not to effect any advantage, unless the heat from the fuel is assisted by the burning sulphur in the ore. If an ore is poor in sulphur, it is useless to build long reverberatory furnaces to roast it in; but if we have an ore carrying say 10 or more per cent. of sulphur, then one is justified in adding another hearth, as the burning sulphur on the first hearth will aid in starting the roasting on the second hearth, and the length of the furnace can be increased with the increase of the percentage of sulphur in the ore.

The ores in such a furnace undergo a gradual heating, and with a furnace having three hearths its conversion into sulphates will take place on the middle hearth, while the chlorination proper will take place on the first hearth near the fire-place.

The ore is fed in at the point which is farthest from the fire-place, and is therefore the coolest part of the furnace. A gradual heating of the ore takes place, preventing it from caking, as the sulphur is gradually driven off, and when it reaches the hotter portions of the hearth there is not sulphur enough left in the ore to cause matting. The ore in being passed from one hearth to the other undergoes a thorough stirring, and as with the same amount of fuel more ore is roasted, the economy of these furnaces over the single hearth is obvious.

The W. H. H. Bowers Furnace.—This furnace has two revolving cylinders, and is so constructed that the ore may be submitted to an oxidizing roasting in the first cylinder and to a chloridizing roasting in the second. The pulp to be roasted is introduced, by means of the elevators, E^1 E^2 (see Figs. 137, 138), hopper A, conveyer B, and spout G^2, in a continuous stream into the back of the calcining furnace F^2. This calcining furnace consists of a cylindrical or tubular barrel, terminating and emptying into the hopper H^2, and is furnished with a suitable grate and fire-box J^2. The cylindrical portion is slightly inclined, as shown in the drawing. The fire on the grate J^2, the speed of rotation, and inclination of the tubular part of the furnace, together with the influx of air through suitable doors or openings above the fire, are so regulated that by the time the ore has arrived at and dropped into the hopper, H^2, it has been roasted until all the sulphurets are decomposed and converted into sulphates and oxides, and has generally been brought into the state best adapted to secure a good chlorination.

The roasted, oxidized, or calcined ore passes from the hopper H^2 of the calcining furnace F^2 through the spout K into the back of the chlorinating furnace F^1 in a continuous stream, while at the same time there is also introduced into the back

of the furnace, by means of the spout, L, and any suitable feeding apparatus, S, a continuous stream of salt, the amount of salt fed being that which is proper for the chlorination of the particular ore which is being roasted.

The construction of the chlorinating furnace F^1 is in all

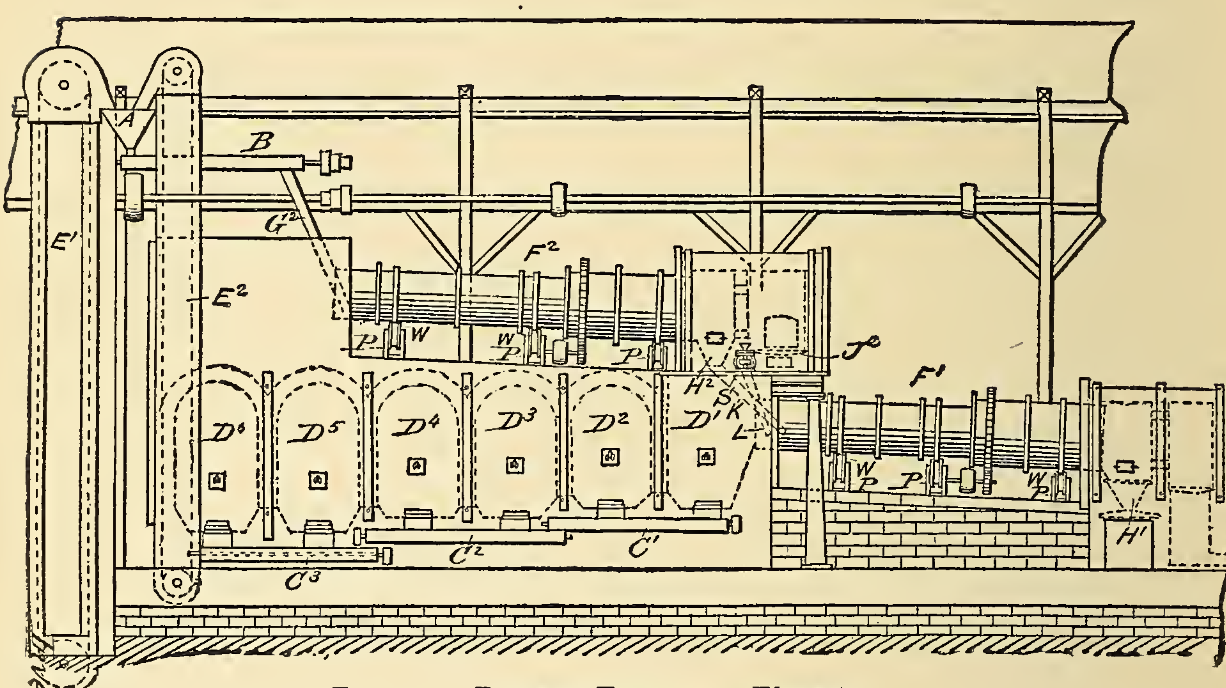


FIG. 137.—BOWERS FURNACE. Elevation.

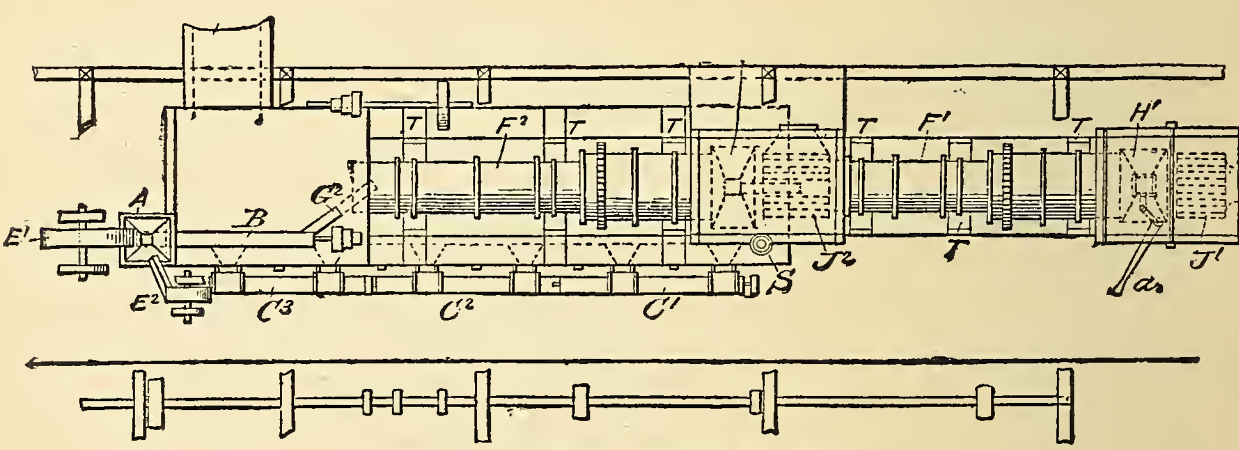


FIG. 138.—BOWERS FURNACE. Plan.

respects similar to that of the calcining furnace F^2, and the parts are similarly lettered. The fire on the grate, J^1, the inclination and speed of the tubular part of the furnace, F^1, and the influx of air are all so regulated that by the time the ore arrives at and falls into the hopper, H^1, it has been fully and properly chlorinated—that is to say, all, or as much as practicable, of

the silver has, together with the smallest possible quantity of the base metals, been converted into chloride.

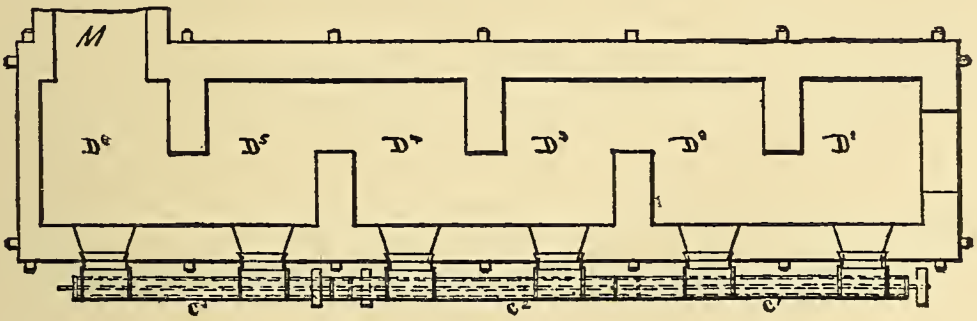


FIG. 139.—BOWERS FURNACE. Plan of Dust Chamber.

Cases may arise in which one chlorinating furnace can chlorinate more ore than can be oxidized by one calcining furnace.

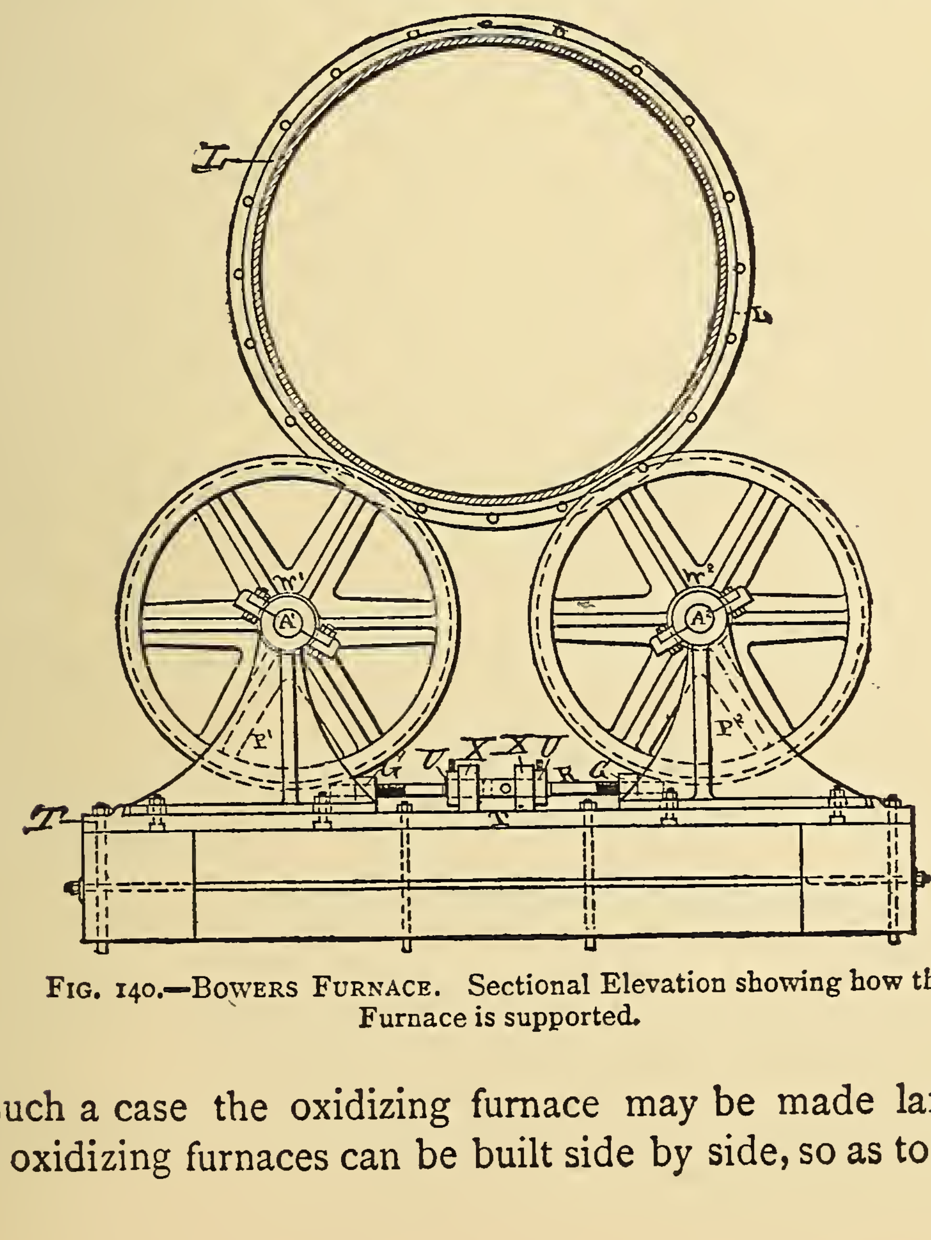


FIG. 140.—BOWERS FURNACE. Sectional Elevation showing how the Furnace is supported.

In such a case the oxidizing furnace may be made larger, or two oxidizing furnaces can be built side by side, so as to deliver

their product by separate or common spouts into the chlorinating furnace.

As the gases from the chlorinating furnace, F^1, will be more or less charged with dust and volatile chlorides, they are passed

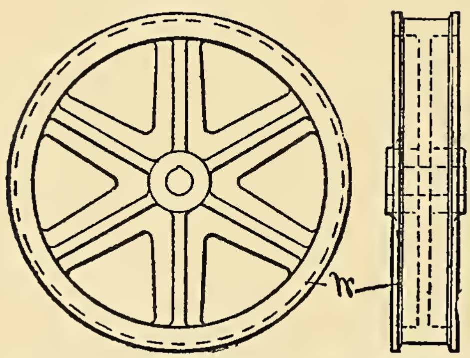


Fig. 141.—Side and Edge View of Driving Wheel.

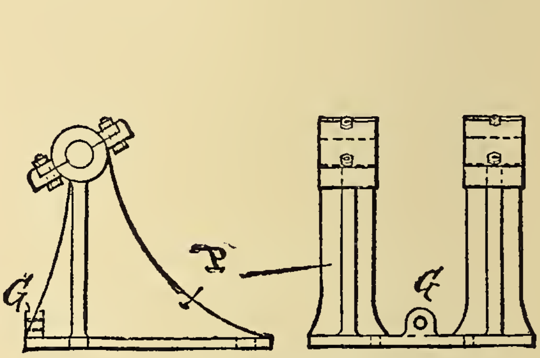


Fig. 142.—Supporting Standards.

Bowers Furnace.

through dust-chambers, D^1 D^2 D^3 D^4 D^5 D^6, and thence into the flue, M, leading to the chimney, and the calcining furnace, F^2, can be built on the top of these dust-chambers, and lead the dust, products of combustion, &c., which it produces, into the

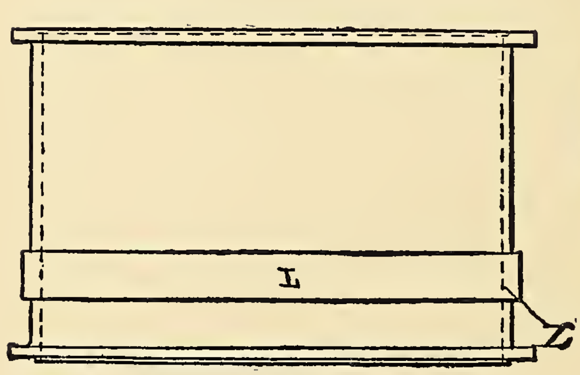


Fig. 143.—Side View of Ring.

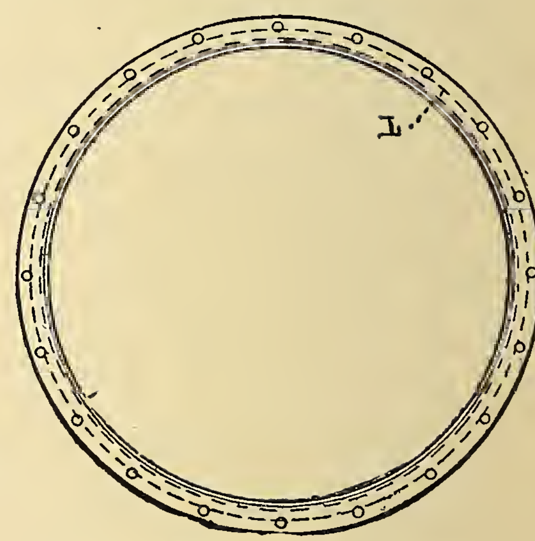


Fig. 144.—End View of Ring.

Bowers Furnace.

same flue, and thence into a common chimney; but the chlorinating furnace does not discharge its fumes into the oxidizing furnace on their way to the chimney.

In constructing the hopper, H^1, the same is fitted with a door or slide to regulate the discharge of the roasted ore, as

in Fig. 138, in which *a* is a handle or lever to move the slide in the bottom of the hopper. Some of the rings which compose the furnace-barrels have cast upon them the projecting tracks, L, Figs. 140, 143, 144. These tracks run in two bearing-wheels, W^1 W^2 (Figs. 140, 141), which are carried by shafts, A^1 A^2, journalled in the standards, P^1 and P^2. These standards rest on the ribbed or grooved bed plate, T, Fig. 140, and are united by a capstan-screw, R, that is cut with a right-hand screw at one end and a left-hand screw at the other. This capstan-screw passes through lugs, X X, on the bed plate, T, and is secured in place by set-collars, U U, as shown in Fig. 140. The threaded portions of the screw, R, take into the tapped lugs, G, on the standards, P^1 and P^2, and it is evident that by turning the screw, R, the bearing-wheels, W^1 and W^2, can be moved nearer together or further apart, and thus the alignment of the furnace altered.

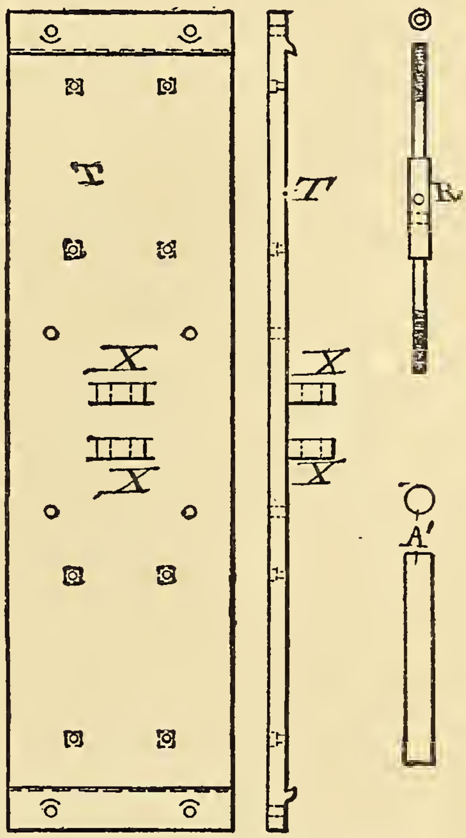


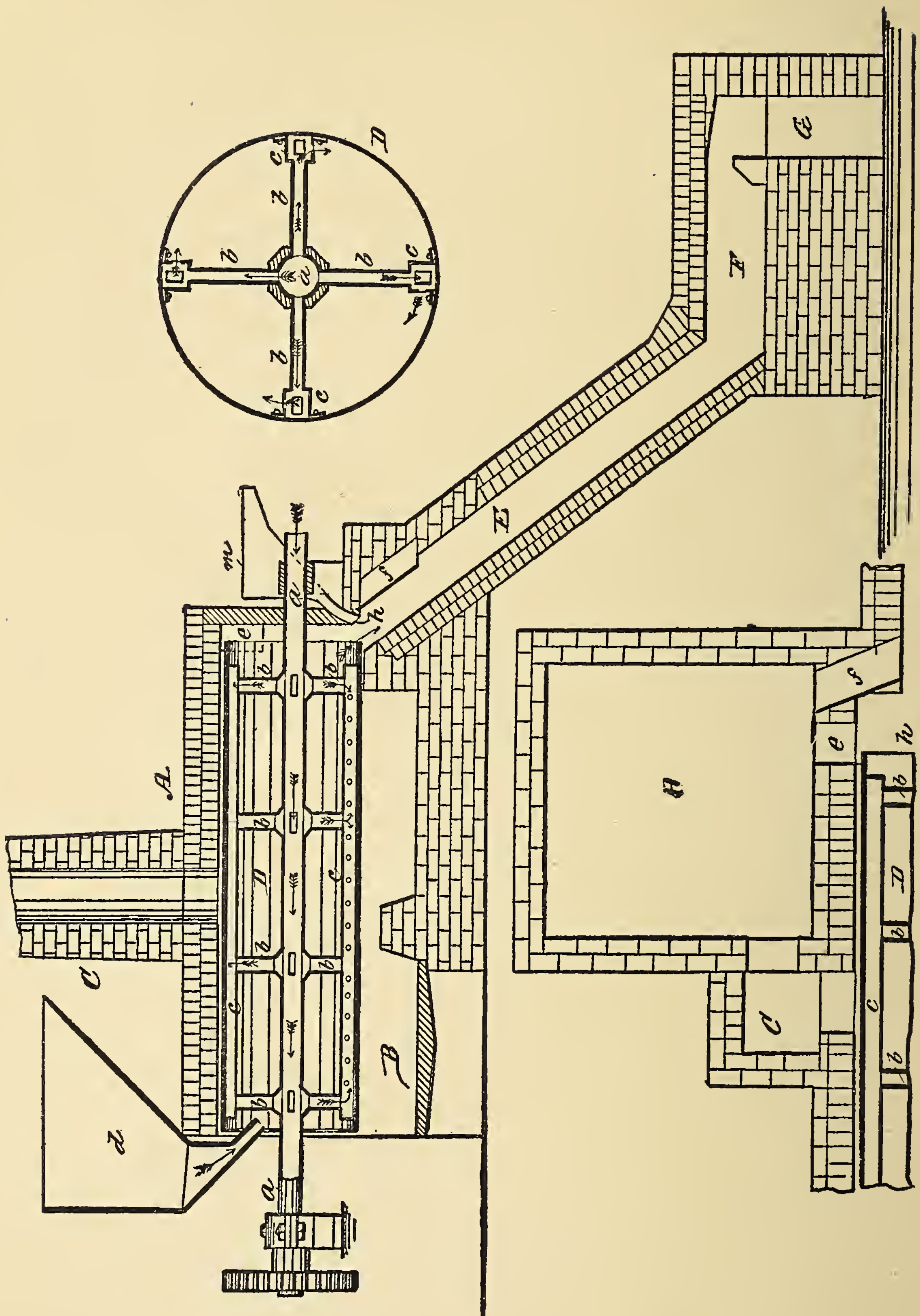
FIG. 145.—Plan and Edge View of Bed Plate. FIGS. 146, 147.—Details.

BOWERS FURNACE.

The R. L. Thompson Furnace.—Fig. 148 is a vertical section of the roasting cylinder of this furnace; Fig. 149 a cross-section of the cylinder; and Fig. 150 a partial plan view, showing the settling chambers.

A is the roasting furnace, formed with fire-box, B, and with a smoke stack, C. D is a hollow cylinder, fitted within the furnace A upon a hollow axle, *a*, that is supported on suitable bearings at its ends, and provided with a gear wheel for connection with the power to revolve the cylinder.

The axle, *a*, is provided with hollow arms, *b*, which connect at their outer ends with tubes, *c*, running lengthwise of the



FIGS. 148, 149, 150.—R. L. THOMPSON FURNACE.

cylinder, upon which tubes the shell is secured. These lengthwise tubes, *c*, are made of rectangular form in cross section, so that they project as shelves or lifts from the inner surface of

the cylinder, for raising the ore in its passage through. *d* is the feed hopper supplying the ore to the cylinder.

E is an inclined flue or chute, extending from the rear end of the furnace, A, at the end of the cylinder, D, downward at a steep incline to the smelting oven or hearth, F, of a second furnace, G. At the rear end of the roasting furnace, A, is a flue, *e*, extending into the settling-chamber, H, and at the upper end of the inclined flue, E, at the point where the ore is discharged from the cylinder into the chute, is a flue, *f*, also extending to the settling-chamber, H.

The ore is fed from the hopper, *d*, to the cylinder, D, which is heated externally by the furnace, A., The cylinder is supplied with air, which is forced by a blower or other means from the outside through the hollow axle, *a*, and passes through the hollow arms, *b*, to the longitudinal tubes, *c*, which are perforated as shown, to allow escape of the air into the cylinder in contact with the ore. In this passage of the air through the hollow axle and arms it becomes partially heated by contact with the heated tubes, thereby effecting economy in the fuel required for roasting the ore.

At the same time the air by its absorption of heat from the axle and arms of the cylinder, keeps those parts from becoming excessively heated, and increases their durability. As the cylinder revolves the ore is lifted by the tubes, *c*, and carried to the upper side, from whence it drops again to the bottom of the cylinder, and it is thus worked forward and finally discharged from the end of the cylinder into the flue, E, through which it passes in a thin stratum exposed to the action of the flame from the furnace, G, and is finally received upon the hearth, F, on which it is permitted to lie a suitable length of time, according to the kind of ore that is being worked, before being raked out upon the cooling-hearth.

In case chloridizing of the ore is required, salt is to be introduced by a continuous-feeding device at the point marked *h*, at the upper end of the chute E, where the hot ore leaves the cylinder D and enters the inclined flue. At this point the salt is rapidly decomposed and utilised with the greatest economy

in the chloridizing of the ore. The chlorine gases and other products of the decomposition pass by the flue, *f*, directly through the settling chamber, H, so that they do not come in contact with and destroy the iron-work of the furnace. The sulphurous gases and other products of the roasting process in the cylinder are taken directly by the flue, *e*, into the settling chamber, while the smoke and gases from the fire-box, B, pass out by the smoke-stack, C, thereby lessening the volume of gas passing through the settling chamber, and to that extent decreasing the current in the chamber and hastening the settling process. The draught of the furnace, G, is to be regulated so as to expose the ore to a carbonaceous or reducing flame, for the purpose of decomposing the sulphates present in the ore, or for the admission of a greater quantity of air, as required in chloridizing the ore.

Comportment of Other Metals in the Amalgamation of Silver Ores.— The metals which are usually associated with silver and auriferous silver ores are: copper, lead, zinc, iron, antimony, bismuth, arsenic, and manganese, and some remarks as to their comportment during amalgamation will not be out of place here.

Copper.—That copper forms an amalgam with mercury is well known, but the exact function of this metal during pan amalgamation I have never been able to determine exactly. Some German authorities hold that when freshly precipitated copper from its salts comes in contact with mercury it will form very easily an amalgam; and in corroboration of this statement I may mention that I have used in pan amalgamation of ores entirely free from copper a very large excess of copper sulphate far beyond what would be required to react on chloride of sodium, but the bullion produced was perfectly free from copper, and the strained quicksilver did not show any copper reactions.

As shown in previous chapters, ores which are subjected to a chloridizing roasting, like those at Mineral Hill, will give a bullion about five hundred to six hundred fine, the other

portion being mainly copper, showing that the cupric and cuprous chlorides formed during roasting are most likely reduced by the iron in the pans; metallic copper is precipitated, which combines with mercury, and what goes in the tailings is either a copper oxide or an oxychloride. If there is much lime in the ore, the tendency will be to carry the copper as an oxide into the tailings, and the bullion will be purer.

I doubt very much whether silver ores carrying even a considerable percentage of oxidized copper produce a cupriferous bullion in the Washoe process; at least, it is contrary to my experience; but my experience taught me, again, that the presence of copper amalgam in the quicksilver is highly beneficial in silver extraction, as the baser the bullion I produced the higher was the percentage of silver extraction.

Lead.—Metallic lead has a great affinity for quicksilver, and requires for its saturation about one-half the quantity of quicksilver as does silver. No heat or agitation is necessary for its amalgamation. It forms a sticky, greasy amalgam, which is covered with a film of suboxide of lead; and lead, therefore, is not a desirable substance in amalgamation. In a previous chapter it has been shown that the presence of very large quantities of lead ores will not interfere with the amalgamation process.*

Zinc, which occurs very often in ores of silver, does not enter into the amalgam during amalgamation, as such ores are usually roasted. The zinc is then driven off, and the balance is converted into a soluble salt, which is not precipitated by iron. Metallic zinc forms amalgam very easily with quicksilver.

Iron.—I have already shown (page 313) that iron forms an amalgam with quicksilver, but under what conditions I am not able to define very clearly. Some electro-galvanic agencies must be at work to cause such a union, and no doubt further researches on the subject are necessary. Iron is one of the prime motors in furthering the amalgamation of silver. It

* See under the heading "Milling Base Ores at Piache, Nevada," *ante*, pp. 130—132.

attracts both gold and silver amalgam, which cover the iron in pans with such hard layers, that the use of chisel and hammer is required to detach it. The regularity with which these amalgam layers deposit on iron suggests the idea that electro-galvanic agency is at work.

Antimony.—This metal amalgamates with quicksilver with difficulty, and as antimonial ores are generally roasted, it plays no important part during amalgamation.

Arsenic, also, amalgamates with difficulty. Both arsenical and antimonial ores have to be roasted previous to amalgamation, as otherwise they would not give up the silver. Arsenic is a bad metal to get into the pans; it cuts up the quicksilver, converting it into a suboxide, which forms a coating over the globules, and these are lost in the wash-water. During roasting, arsenic will decompose chloride of sodium, causing an evolution of chlorine, which is an advantage in the absence of pyritic ores.

Manganese, according to observations I have made on roasted ores, when discharged into the settler and water added, causes a thick froth to be formed, which carries fine particles of quicksilver in suspension.

Among the earthy ingredients which form the gangue of silver ores, are to be found, besides quartz, clay, baryta, and lime. It is well known that silver ores with a quartzose gangue, both in wet and dry milling, give better results than when a large percentage of earthy minerals is mixed with them.

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